METALLURGY

OF

SILVER, GOLD, AND MERCURY

IN THE

UNITED STATES.

THE

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BY

THOMAS EGLESTON, LL.D.

IN TWO VOLUMES.

VOL. I.

SILVER.

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TO

LOUIS GRUNER,

INSPECTOR-GENERAL OF MINES OF FRANCE;

A LEARNED PROFESSOR,

A GREAT INVESTIGATOR,

AND

A DISTINGUISHED SCHOLAR,

THESE VOLUMES ARE GRATEFULLY INSCRIBED

BY

HIS PUPIL,

THE AUTHOR.

PREFACE.

THESE volumes were commenced in an endeavour to present to my pupils a systematic and comparative view of the different processes used for the extraction of gold, silver, and mercury in the United States. Since their commencement many years ago, the work has grown far beyond the space that it was originally intended that it should occupy, and I have ventured to present it to the public in the hope that it might give, up to this date, a reliable description of the condition of the metallurgy of gold, silver, and mercury in the United States, as well as the progress which has been made towards perfecting machinery and processes for their extraction. Much of the information contained in these volumes has been published from time to time, as the examinations or researches were made, in the various scientific periodicals in this country and Europe, as detached and disconnected memoirs and monographs; these, with much additional material, have now been put together in such a way as to represent the treatment of the precious metals in the United States as a whole. The investigations on the various processes and the researches on such subjects as seemed at the time to require special study, have been made either at the works, through the kindness of the superintendents, or in my own

laboratory, simply for the sake of ascertaining, as far as was possible, what the real state of the facts were. As stated in the Introduction, the progress has been extremely rapid, so much so that processes, which have been in the past of the highest importance, have been entirely superseded by others, and sometimes within a very few months, ores, which it had been found previously impossible to work, either on account of the complication of the impurities contained in them, or the poverty of their yield, have become possible to treat with perfect success. There will undoubtedly be as great progress made in the next ten or twenty years as there has been in the last two decades. Our surface ores, which are in general easily treated, have been largely worked out, and we are now coming to the period when we shall have to attack the real wealth of the country, which is contained in the poor and the so-called "rebellious ores," of which there are and always have been plenty. The use of this word rebellious has been the cause of the loss of many millions of dollars. It is one of those words behind which men are willing to attempt to conceal great ignorance, and in the face of which they have been content to use the metallurgical *non possumus*. If, instead of using this word, it had long ago been said that the ores now known as rebellious are those which we do not now know how to treat, men would not have ascribed an active principle to the ore, as though it was resisting them, but might have constantly worked at the problem of treating it, until some successful method had been found. The spirit of to-day is to recognise no difficulty which, in the face of transportation, fuel, and water, may not be overcome. By improved and more economical methods of treatment, the precious metal contents at which the ore is workable, is steadily

becoming lower, so that ores which could not be worked twenty years ago, now yield a profit. Our progress in the past has consisted in eliminating many of the formerly poor and rebellious ores from the list, and it is to be hoped that our progress in the future will consist in eliminating rebellious ores altogether, so that there will be no class of ores unprofitable to treat on account of their constitution, but only on account of their yield. The commercial details of the various processes have been given as far as it has been possible to obtain them. There is, however, a natural reluctance to give these details, as such information often leads to competition, and frequently to losses. These details, while they are generally not complete, for individual works, give, in most cases, a very fair idea of the cost, and are generally sufficiently full to be made the basis of estimates.

I have used for the most part, in writing these volumes, my own notes taken at the works, but I have also consulted freely the Transactions of the Institute of Mining Engineers from 1871 to 1886, the Reports of the Mining Commissioner from 1867 to 1876, the Production of Gold and Silver in the United States from 1880 to 1884, and the thirteenth volume of the Reports of the United States census of 1880. To individuals I am especially indebted to Mr. Faber Du Faur for working drawings of his furnaces, to Mr. Weisse, of the Germania Works, and to Mr. Eurich, of the Pennsylvania Lead Works, for many interesting details on zinc desilverization furnished to me by them, while visiting their works for the preparation of that chapter, to Mr. A. Eilers, of South Pueblo, Colorado, and to Mr. Malvern Iles, of Denver, for information regarding the present prices of ores; for information relating to the Boston

and Colorado process, to Professor Hill, who afforded me every facility for taking the drawings of all the furnaces, and to Professor Pearce, who gave me the information required at the works, concerning the details of the various processes carried out there; for information relating to silver stamp mills, to the Blake Crusher Company, of New Haven, Conn., for information and drawings, to Mr. S. R. Krom, for drawings and information, to Mr. J. M. Scott, of the Union Iron Works, and to the proprietors of the Miners' Foundry of San Francisco, who kindly allowed me to take tracings of drawings and to examine the mills which they have constructed; for information on pan amalgamation to the Miners' Foundry of San Francisco for drawings, to Fraser and Chalmers, of Chicago, Ill., for drawings and information, to Mr. W. H. Patton, the designer and constructor of the Consolidated Virginia Mill for the loan of, and the permission to copy, the drawings from which that mill was built, to Mr. Elstner, of the Brunswick Mill, who furnished me, both at the mill and by letter, with a large amount of information, and to the managers of the mills generally who have very kindly entered into my plan of endeavouring to present a fair outline of the methods in use on the western coast for the milling of silver ores; for the treatment of tails, to Mr. W. C. Armstrong for the drawings of the Woodworth Sluices, to Mr. Hodges, of the Lyon Mill, for information concerning that mill, and to my former pupil, Mr. J. A. Church, for information relating to the Tombstone Mill. For the processes of leaching silver, to Mr. J. H. Russell, and to my former pupil, Mr. C. F. Pearis. For information relating to the Mabel Mine, to my former pupil, Mr. W. Radford. For information on gold stamps, to Mr. W. H. Patton for the loan of and permission to copy

the construction drawings of the Plumas Eureka Mill, to Mr. O. C. Kensett, superintendent of the Keystone Mill, for information given at the time of my visit, and to Mr. G. F. Van Deetken for some details on blanket washing; for information on the parting processes used in the United States Mint, to Mr. H. Burchard, the former Director of the Mint at Washington, and to the present Director, Dr. J. Kimball, who have given me the necessary authority to visit the Mints and Assay Offices of the United States, and also to Mr. A. Mason, Superintendent of the New York Assay Office, and Mr. B. F. Martin, the melter and refiner; for facilities in studying the process used there, to Dr. Booth, of the United States Mint, at Philadelphia, and especially to Mr. W. G. Summer, Mr. F. C. Garrigus, and Mr. D. H. Mirkel, for the special courtesies and information given me while visiting the Mint to study the processes there used, and to Mr. J. F. Randoll, of New Almaden, to Messrs. C. E. Livermore and J. W. Hall, of the Reddington Mine, and also to Mr. R. F. Knox, of San Francisco, for information relating to the treatment of mercury.

School of Mines,
New York, *July*, 1887.

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INTRODUCTION.

THE progress of metallurgy in the United States during the last thirty years has not only been very rapid, but has produced very extraordinary results. Previous to the discovery of gold in California in 1848, the United States produced but a small portion of the world's product of metals, and its people were not known as skilled in the mining or metallurgical sciences. The great coal-fields were known and worked, and iron was produced in what would now be considered very small quantities. The discovery of copper on the upper peninsula of Lake Superior, and of gold in California, showed the want of knowledge of these subjects amongst our people, and undoubtedly gave the stimulus to the energy and enterprise which now makes the United States one of the greatest of mining and metallurgical nations. Twenty-five years ago there was no place in the United States where an elementary knowledge of either mining or metallurgy could be had in any of the educational institutions of the country. To-day a more thorough knowledge of these sciences can be had here than anywhere else. The prospector of

those early days has been obliged to give way to the thoroughly trained mining and metallurgical engineer. No finer or better examples of what skill and science can do are to be found than in the works which have been erected in different sections of the United States within the last fifteen years. In the early days, when prospectors and mineral wands were more or less relied on, "science," which meant education in general, was looked on with suspicion; the demand was for "practical men." But as soon as it became apparent that there was enormous wealth in the ground, which could not be extracted by either uneducated or half-educated men with empirical means, science was called in. In the first few years the "scientific cuss," as he was called, was expected to be a mining, metallurgical, mechanical, and chemical engineer, and in all cases where he did not know, to improvise his knowledge; but it was not long before experts in all the branches were to be had, and, in consequence, new methods and new processes have resulted, so that mining and metallurgy have made more progress on this side of the Atlantic in the last thirty years, than had formerly been made in the Old World in more than one hundred. A quality of mind seems to have been produced by the mixture of races on this continent, which, while demanding for the people as a whole the most general information, enables us easily to grasp details, while the eminently mechanical genius of our people leads us to adapt, improve, and invent machinery for every purpose, and thus, by replacing hand labour, to both improve and cheapen the cost of processes.

The gold excitement in 1848 was produced by the discovery of the very rich placer mines at Sutters Mill, El Dorado Co., California, from which the gold in quite large pieces was easily separated by washing with water in a common pan. Shortly after, the discovery of a number of nuggets of large size, and the rumour that others had been found and cut into small pieces to prevent their value being known, caused the wildest enthusiasm all over the United States. Some large fortunes were made among the crowd of adventurers who rushed to California, but no one can know of the misery and disappointment caused by this wholesale emigration. The success of the few was widely published, but the record of the failures was never made. What

they must have been is shown by the fact that after the State of California had provided insane asylums for the percentage usual in other States, the authorities were obliged to double their capacity.*

So entirely has mining formed a part of the history of the country, that the very words which express the miner's joy or sorrow, his success or failure, have passed into the current conversation of society. To "strike it rich," to "find pay," to "pan out," or to "get down to hard-pan" or "to the bed-rock," indicate to the people of the Western coast their success or failure, their prosperity or adversity, more emphatically than any other words in the English language could do.

It is very doubtful whether the Western States and Territories would have been settled even now but for the fabulous yield of the early-discovered surface deposits. California had been more or less occupied for 200 years by Spaniards and Mexicans. It was known in a general way that gold was to be found either in that country or districts adjacent to it. Shallow placers had been worked with very small results for a long time. Placer gold was discovered and worked in 1841.† In 1843 gold-bearing quartz had been mined. But the discovery of gold in the sluice of a mill depopulated, for the time being, the city of San Francisco, brought a small population from China, and thousands from Mexico; but the country was in a very few months taken possession of by men from the Eastern States, attracted there both by the fabulous stories about the gold and by their love of adventure. Everybody who could dig (and who could not?) became at once a miner. It was not an unfrequent thing, so rich were the deposits, for men who could not earn more than 75 cents a day where they came from, to make several hundred dollars between sunrise and sunset, with the rudest appliances, and with a loss of no one knows how much gold beside. This great gain was, however, of but little use to him, because, having acquired it without much labour, no special value was attached to it, and he spent it as freely as he found it. The price of

* I have this statement from a former Treasurer of the State of California.

† "Engineering and Mining Journal," vol. xxxii., p. 170.

everything in California was, consequently, very high.* Trade was carried on as barter, in which "gold dust," as it was called, which was of all sizes, from a nugget the size of the fist to an almost impalpable powder, answered the purpose of a medium of exchange. In every-day life the fine material was used in the place of coin. It was carried by the miners in buckskin bags which were opened for the merchant to take his own pay by repeatedly taking out what he could hold between his thumb and forefinger. A single pinch from the bag was considered to be equal to 25 cents, which was the lowest value recognised; two pinches were equivalent to 50 cents, four to $1, and so on. The honest (?) merchant of those days kept his forefinger pressed on a conical button in his waistcoat, in order to make a depression in it, so that the intrinsic value of the single pinch was often equal to a dollar, or even more. This was of no consequence whatever to the miner, who did not condescend to weigh his gold, except in the case of nuggets or of large payments or bank deposits.

At first, only the shallow and very rich placer mines were worked. They required no capital, and but an extremely limited amount of intelligence and skill to work them. So long as they were easily found and the pay abundant, the separation of the gold was only approximative, and no special improvement was made in the processes of extraction. If the sand was very rich and water was not easy to be had, it was simply winnowed in a blanket, and what the wind did not blow off was amalgamated when mercury could be procured and a partial extraction effected, or it was panned in a rough way when water could be had, or was approximately picked out when there was neither water nor mercury. Any method was used without regard to loss, so long as the "pay" was made rich enough to transport, and could be sold by assay.

By such rude methods only a very small percentage of the gold was extracted, but it would cost trouble to get it all out, sc it was left where it was. The miner was then careless of his

* Flour, $7.50 per pound; butter, the same; a barrel of pork $210; a bottle of ale, $5; a candle, $3; a tin pan, $9.

mode of working, giving no thought to the future, and was extravagant in his habits and methods. In less than five years after its first discovery, the quantity of gold had reached a maximum, and after 1853 began to decrease steadily. As the amount of gold diminished the methods of extracting it improved; the miners became aware that they were losing gold, and little by little they ascertained that this loss occurred in several different ways: either it was not separated from the lumps of sand or clay and was carried off with them, or it was so thin in the pay rock or became so thin in the process of extraction that it was buoyed up and carried off by the water, or was lost because the gold for some reason did not amalgamate. At first he simply called this light gold "float" and the other "rusty," and made no very great account of it. But as it became evident that a very large proportion of the gold in the "pay" was being lost in this float and rusty gold, he became anxious first to ascertain how much there was that he did not catch and what part of this was float—a material so light that it was carried off upon the water, and was thus swept away—and what part was rusty, that is, did not amalgamate with the mercury. To determine this he sought for machines to work with. The miner's pan had up to this time been the tool with which he "got the colour," which is the expression used when the gold begins to appear after washing the pay dirt in a pan; but when he found that he must have other appliances, the miner of 1848 took the processes which he found in use by the Mexicans and the Mexican Indians in California, and adopted them bodily. These were, the miner's pan, which he found in use when he first came into the country, and continued to use so long as the placers were very rich; the arastra, and the Chilian mill upon which so much ridicule has been cast in later times, and to the principle of which we shall probably have to come back as the gold grows scarcer. The rude methods of concentration associated with these machines were the only methods that he found there, and these he accepted without at first thinking of any improvement.

With the altered condition of affairs every experiment that want of experience or an active imagination could suggest was

tried. Every man improvised himself a miner, and felt it his bounden duty to commence where Tubal Cain did, but without his intelligence. The sum of all that is left of these experiments is the sluices, riffles, and undercurrents which are still in use. They alone represent the activity and energy which in the first few years produced hundreds of failures, and brought loss and disappointment, and in many cases ruin, to thousands of men. But the necessities of the case had so stimulated the active minds who brought their energies to bear on the subject that the failures were individual, as the loss to a country of wealth which no one owned was not estimated, and, consequently, was not felt. The country reaped the benefit of all these efforts and all the gains, and in consequence has made greater strides in the way of progress, and more improvement in thirty years than has ever been made in a century by any other nation.

It is impossible to estimate the losses which must have occurred in these early days by such rough methods of working. The "Forty-niners," as they are called, cared very little whether they got out all the gold. What they did care for was to make the greatest present profit not only between the first and last of the month, but between sunrise and sunset of every day. Sunday did not count for a holiday then. It was of no consequence to the miner what he lost, but it was of great importance to him what he gained. He recognised only to-day; he knew and cared nothing about the time when through his slovenly methods gold would become scarce, and if he knew that he was losing gold his regret was for himself because he did not get it, and not that the country was a loser. He knew nothing about political economy, and would have cared little for it if he had. He was extravagant in his habits, generous to a fault, careless of his methods, entirely satisfied to let the future take care of itself. By the time that the placers began to grow less productive the population of the country had increased to such an extent that labour becoming abundant and wages so comparatively low, the necessity of living, which Talleyrand declared not to be necessary, became a stimulus towards improvement in the processes and to the effort to save what had been previously considered as not worth saving. More careful washing made men

look for a more abundant water supply. The freer use of mercury enhanced its value and set the prospectors searching for mercury mines.*

Our present methods, which are being constantly improved, appear extravagant to those who have been trained in the European schools, because they do not take into consideration the fact that with a rate of wages much higher than in Europe, and with transportation at its maximum, their economies would cost more to carry out than they would yield. Hence the foreigner coming to this country has generally to commence his education here by unlearning some parts of the knowledge which he has been taught abroad to consider as his first principles, before he begins to acquire the new ideas which are essential to his successful practice among our mines and metallurgical works.

There is in this country no impediment to progress and improvement, as there is in Europe, in what is known as ancient legislation, where antiquated processes have become customs, and their use precedents not to be easily set aside. Their statute books are full of laws preserving the rights of the government, of John Doe and Richard Roe, in certain specific things, and requiring, in certain defined cases, the following of ancient precedents or the use of obsolete processes and methods, the laws being made with reference to a condition of things which existed perhaps one hundred, three hundred, or even five hundred years ago, and remaining still unrepealed. It is undoubtedly in principle a wise precaution not to allow the investment of capital in mining enterprises without having the whole question, both in a commercial and industrial point of view, examined by government officers, and the government sanction given or withheld according as, in the judgment of its experienced officers, it should or should not be given; but this kind of paternal government places a great deal of power in the hands of individuals who desire to hold the leading-strings, and does away with the individuality of enterprise which is one of the chief characteristics

* The discovery of very large deposits of the ores of mercury in California, and the original processes invented there for working it, are quite as remarkable as the progress made in the working of the other metals.

of our nation. The anxiety to prevent the individual loss in Europe often retards the prosperity of a whole country, while the precedents, as well as the written and unwritten laws of some of the older countries, and the system of centralisation in their governments which entrusts everything belonging to the government to corps or to the heads of bureaus at the seat of government, are great impediments to progress. In this country we are at liberty to deal with the necessities of the case as they occur. The impediments to progress are not too much but too little law, not ancient but conflicting legislation. These impediments are, however, incident to any new order of things, and it will not be many years before the last of these difficulties will be removed.

When shallow placer-mining became less remunerative, the question of transportation commenced to be a factor in the question of production. While a miner was earning $100 a day, the fact that he was forced to "pack" every article he purchased for use, as well as his gold, on a mule to be carried for days or weeks, or concealed his treasure about his person and travelled on foot a considerable distance to invest or deposit his dust was a matter of very little consequence in view of the large amount so easily earned.*

But when he was reduced to "hard-pan" and only earned a little more than was necessary for the sustenance of life, the question both of improved processes and of transportation became a very serious one, and had to be studied very carefully. At this point he either improvised himself an engineer of mines, and with ready Yankee wit devised some process which would save "the colour," or transported himself and his "outfit," which in those days meant what he would carry on his back, with his pick, spade, pan, horn or spoon, or more probably all of them, to some locality which was not so far removed from civilisation.

* The safety and almost absolute impunity with which large amounts were carried about the person in the early days of California mining was owing to the rapidity with which the Vigilance Committees of those days executed justice, which was not only speedy but severe. There were then no lawyers to impede the progress of what was almost invariably real justice, though it was often not good law, and death was usually the penalty inflicted.

Others, on the contrary, sought, in localities farther from civilisation, richer pay diggings.

It was not long before this continued migration produced its results, for while the miner at first searched only for gold and worked for gold alone, when it became scarce and the necessity of living became to him imperative, translating itself into the gnawings of hunger, other things became valuable besides gold, and he then stopped in his transit from one district to another in search of the precious metal to locate a silver deposit, sometimes in the shape of lead ores, but he never noticed iron or copper. It was not until long after that he made the discovery that copper ores frequently carry gold, and lead ores both gold and silver, and then he sought for smelting processes. This was not done, however, until the placers had commenced to decrease materially in their yield.

PLACER MINING.

The deposits of gold first worked were called placers. These were from the commencement recognised as dry and wet, the dry being either ancient river deposits or recent river beds from which the water was gone. The wet placers were streams which were sometimes large rivers. In these placers the gold was within 20 ft. or so of the surface. The dry placers were worked with pans and sluices; the rivers containing the wet placers were dammed and the whole of the water turned into flumes or sluices. The river mud was then sluiced. As it was impossible to keep the river bed dry, both because the dam leaked and because the surface drainage could not be kept out, large wheels called "flutter wheels" were placed in, and worked by, the artificial channels, which pumped the water out of the river bed and furnished a sufficient supply of it to the sluices. Such workings were hazardous and expensive, for if the stream was at all large, a single freshet might wash out the dam and the flume which carried the river. To make sufficient room to work and also to get enough ground to justify the expense, the flume carrying the water of the river was never less than 250 ft., and sometimes

several times that length. Such works were very expensive and somewhat hazardous, but sometimes yielded a very large profit. They were constructed by companies during the early days when the miners used less engineering skill and were willing to take more risks. Very few of these river claims exist now, and this kind of mining is already a thing of the past.

As the placers grew poorer the gold was sought deeper, and it was then discovered that there were placers where the depth of the deposit was over 300 ft., where the gold on the surface sometimes yielded only a few cents to the cubic yard, but was rich on the bed-rock, hence the name of deep placers. Only shallow or surface placers and deep placers are now recognised. It is estimated that two-thirds of all the gold produced in the world is taken from placer deposits.

Long before the fact that the gold was decreasing became known to the miners, it was evident that some new system would have to be used to extract it. The miner's pan, a household utensil, was used, not because it was the best adapted to the work, but because it was the most available. It was an exceedingly rude apparatus. It was made of sheet-iron from 4 in. to 5 in. deep. It was filled two-thirds full of dirt, and put into a hole about 1 ft. deep filled with water, and the contents of the pan well stirred with the hands; the pan was then taken in both hands, inclined slightly outward, and shaken so as to give a circular motion to its contents. The fine material flows, in this way, over the sides. When this has been done long enough, the stones are removed, any lumps of clay are broken with the hand; then, depressing the outer edge, it is shaken until nearly everything but the black sand is out of the pan. This is separated by blowing. It required great skill, and the results showed a large loss in gold. It is now rarely used except for an assay for colour, for which the shovel can also be used. The pan was used for a long time after other machines were introduced, but it soon became necessary to have other implements. As an instrument for concentration, it was succeeded by the cradle or rocker, which was not unlike a child's cradle, whose maximum capacity was five to six cubic yards of earth a day. It required two men to work this cradle efficiently, one to rock and pour on

water, and one to bring the pay dirt and the water, whose weight was at least three times that of the dirt. The pay dirt was thrown upon a screen whose object was to separate the large stones and to help break up the clay; the purpose of the water was to take up the finely divided particles. The concentrates were then panned. A pan contains about half a cubic foot of pay dirt. A one-man rocker could concentrate from 100 to 150 pans a day, and a two-man rocker twice as much. It is a very slow machine, must always be placed near the water, no matter what the distance of the pay dirt is, and even when used with quicksilver loses a very large amount of fine gold and all the float gold.

The cradle was succeeded by the "long Tom," which for nearly two years seemed a great improvement. It consists of a rough trough or launder 18 in. wide at the upper end and 30 in. at the lower, and 12 ft. long. The lower end is terminated by a screen of iron whose edge is so high that the water does not flow over it, but drops with its contents into a trough below provided with riffle bars and mercury. The pay is thrown in at the upper end and mixed with water; what passes the screens comes in contact with the mercury and is caught, but much of the fine and all the float gold is lost, though it is a better machine than the cradle. As many as four men can work at a Tom, but it is now rarely seen.

At the same time a puddler was used, which was a barrel cut in half or a rough wooden box 6 ft. to 8 ft. square, and 12 in. high, with 1½ in. hole 4 in. from the bottom. The pay was thrown into this and broken up and mixed with the water until it was all in suspension. When the plug was removed, the thin mud was allowed to run out and the operation commenced again, and so on. The gold was collected in the bottom. The puddler never was used in this country except in small claims where water was scarce, but has been very successfully and extensively used in other countries where water was not abundant.

The next improvement was the sluice, which, although it had been formerly used elsewhere, is as much a California invention as if it had never been used before it was invented here. It consists of a trough or launder made of rough 1½ in. boards sawn

for the purpose, and nailed together without any pretence of making them fit, as it becomes tight by the swelling of the wood and the filling up of the cracks. The sluice is composed of a series of boxes, as they are called; each box is 12 ft. long, 16 in. to 18 in. wide at one end, and 20 in. to 24 in. at the other. The height of the sides varies from 8 in. to 24 in., according to the kind of material to be used in it. The narrow end of one box fits into the wide end of the other. The length of the sluice is estimated in boxes; it should not be less than 50 ft. long, and often consists of hundreds of boxes put together supported on rough trestles. The inclination is regulated according to the necessities of the case, and is called the "grade." It is given in inches, as "a 12-in. grade," which always has reference to the box. It is usually 8 in. for a minimum and 30 in. for a maximum in 12 ft., or the length of a box. It is generally uniform, but sometimes is made to conform somewhat to the lay of the ground. The tougher the dirt, the longer and steeper the sluice must be. With a rapid descent the dirt is also much more easily separated, but the greater is the loss in gold. The boards are not only rough on the bottom and sides, but to prevent too rapid wear, and also to help to catch the gold, strips of wood are fastened in the bottom of the box, either in the direction of its length, which is most usual, or across it. These pieces of wood are from 2 in. to 4 in. thick, and from 4 in. to 6 in. wide. When longitudinal they are 6 ft. long, so that two sets go in a sluice. The riffles are wedged in so that they can be easily removed for a clean-up or for repairs. The bottom of each box is thus filled with a screen of rectangular cavities which are the width of the distance between the riffles, and have their length and depth. Here the quicksilver which is put in at the head of the sluice rests and catches the gold as it sinks in the more or less rapid current. The water is generally made to run 2 in. deep over the riffles. The pay dirt is thrown in with shovels. The first dirt, which is always poor, goes to fill up the spaces between the riffles The water, rushing over this, washes out the earth and clay, the sand and gravel, and even the stones. No mercury is added until after the sluice has been running about two hours. It is then put in at the head and finds its way down the sluice,

most of it being caught not far from where it is put in. One man can throw in from two to five cubic yards of dirt a day. The number of men that can work depends on the length of the sluice and the lay of the ground. Sometimes when the earth to be washed contains large stones, an undercurrent is used. The end of the sluice is then open. It terminates with a grating of iron bars long enough to allow all the water to pass with the fine material through the grating, but allows the stones to roll out on the ground. The whole of the water and dirt is caught in a short sluice below and discharged into one parallel with the main sluice, which is thus continued. This undercurrent, though sometime used with the ordinary sluice, is an indispensable part of the plant for hydraulic mining. The sluice is simple in construction and use, requires but little outlay of capital, and is very effective. Unlike the outlays usually made in mining, the whole of the plant in use can be readily taken down and transported to another claim when the first is washed out, and set up there as effectively as if it were new. There is one precaution that must be taken with it, which is that lumps of clay must not be allowed to travel along it without being broken up, for they are liable to pick up the gold as they roll and carry it off with them.

Getting the amalgam out of the sluice is called cleaning-up, and the time between one clean-up and another is called a run. The length of a run will depend on the richness of the deposit, but is usually from six days to two weeks, occasionally longer. A clean-up occupies about half a day, and is usually done on Sunday. To do this the water is allowed to run after the dirt is no longer thrown in, until it is quite clear ; six or eight sets of riffle bars are then taken out at the head of the sluice, and the material washed down, while the amalgam is caught at the head of the next riffles. This is taken out. The next set of riffles are then taken out, and so on. The excess of mercury is strained from the amalgam by twisting it in a buckskin bag, and the rest is driven off by heat.

HYDRAULIC MINING.

For many years the sluice was used to work nearly the whole of the placer gold of the country, and it is still the most available way for persons of small capital to treat the shallow placer deposits. It must always be looked upon as a process of great historical interest, for out of it hydraulic mining grew, which is one of the most marvellous achievements of modern engineering skill, to which the State of California is more indebted than to all other inventions of mining and metallurgy put together. There are localities where poor, shallow placers are found, where water is scarce during most of the year, but abundant at certain seasons, and where the grades are heavy. For these placers another sluice was used, known as the ditch or ground sluice. A small ditch is cut through the placer and the water turned into it, the first object being to deepen and enlarge the ditch to the proper size. When this has been accomplished, the banks are pried into the stream. No mercury is used, but cobble-stones are thrown into the bottom of the ditch so that the gold may settle between them. The effort is to concentrate the gold in the dirt and then work it up in a short board sluice.

These two methods, the board and the ground sluice, put together, were the germs of hydraulic mining now so extensively used in all parts of the world, not only for gold mining, but for the removal of dirt and for washing other ores. It appears singular that the name of the man who really invented the most remarkable process of this century should be as lost to history as if he had never existed; but in the struggle for existence, as the shallow placers grew poorer and poorer, and the gold was found at constantly increasing depths, the man was lost sight of, while his work has now been so perfected that it is one of the marvels of the union of modern engineering skill with capital. The process of hydraulic mining was invented in the spring of the year 1852, on the Yankee Jim claim in Placer Co.,

California, where an enterprising miner, finding that he was not making sufficient money, began working his claim with a shallow ditch in the side of a hill leading to an ordinary barrel, from the bottom of which a cowhide hose was carried and discharged by means of a tin pipe against the bank, and he thus became the father of one of the greatest of modern inventions, hydraulic mining.

There is another kind of placer which deserves notice, both because it is interesting in itself and because it has been the source of disappointment and loss to so many. This is the beach gold, which occurs between Point Mendocino in Northern California and the mouth of the Umpqua River in Southern Oregon. The cliffs along the ocean front seem to be the remains of an ancient river deposit. They contain gold, and where washed by the waves often show the shore for miles glittering with it. It is very uncertain, however, for what appears to-day is either washed away or deeply buried in the sand to-morrow. No dependence can be placed on finding in the morning the deposit of the day before, so that all haste is made to carry the sands which are rich enough to some safe place inland to be washed and amalgamated. The beach is very narrow, and when the waves are high they wash against the bank. The gold is washed out with the heavy sand, and as the particles are very fine, it is carried down to near the low-water mark. When the ocean is still, sands of variable richness can be collected, but the waves are often so high as to wash all the sand away to the depth sometimes of 6 ft., and leave the bare rocks exposed. So changing and shifting is the value of the deposit that it has to be examined every day, and the washing of the following day may sometimes be six miles from that of the previous one. Such sands as these must be very rich to make it possible to run the risk of working them and to bear the transportation to fresh water, as salt water is of too high a specific gravity to work with, for which reason the attempt to work these deposits by dredging has not been successful. It has been proposed to bring water from a distance and turn these mines into hydraulic placers washing down the banks, and depending on them for their profit, leaving the beach sands as an

accessory. The tails would not then be a question of importance, but the economic results of such an enterprise would be very doubtful.

After the introduction of sluicing either on a large or small scale, the pan, the cradle, and the rocker were very rapidly abandoned to John Chinaman, who always succeeded in living off the placers that had been abandoned by the "honest miner," who, however, never had any compunction, if he found the Chinamen were tolerably prosperous, in "jumping" their claims and driving poor John off to find some other place for his enterprise and cheap labour.

From very rich shallow placer deposits which when worked in a small way often yielded hundreds of dollars a day, by a gradual decrease stretching over a period of from fifteen to twenty years, the gold became scarcer and scarcer, until now it has become necessary with improved appliances to work in the deep placers, material which contains from 3 cents to $1.25 per cubic yard.*

No better evidence of the progress that has been made in the working of placer claims can be had than the comparison of the cost of working them by the older and by the more recent processes. Adopting four dollars per day as the wages of a skilled miner, the cost of working a cubic yard of gravel as given by Phillips is—

With the pan	$20.00
,, rocker	5.00
,, long Tom...	1.00
,, hydraulic process	.02

In the very early days the capital required by a miner consisted of a pick, shovel, a horn and spoon and a pan, two stout hands, and a valiant heart; but as the placer mines grew poorer this capital was no longer sufficient. It can readily be understood that such capital as was required in the early days did not

* In the spring of 1880 I was informed by the President of the North Bloomfield Mining Company that they were now working with a profit gravel containing only three cents to the cubic yard, when only a few years ago it was considered a marvel to be able to work that which yielded ten cents.

necessitate any permanent location nor any very high order of intelligence for its use, and hence much of the surface was dug over and simply rendered difficult to work further without extracting more than a tithe of the wealth contained in the ground, merely because the miner had no permanent interest in one spot more than another, and because his capital was entirely a rolling and not an invested one. Every man then was in business for himself on the spot where he could find the most pay, but as the gold grew scarcer, experience first, and then intelligence and capital, became factors in the equation, so that the capital required to work any one of these claims became larger and larger, until to-day it is estimated that the plant of the North Bloomfield Mining Company has cost not less than $3,000,000, and these works are marvels both for the originality and engineering ability displayed, consisting of dams 90 ft. to 100 ft. high, ditches, pipes, and sluices many miles in length, and every hydraulic appliance which engineering skill and capital can add to their works.

These deep placers are the deposits formed in the beds of ancient rivers, which have since been so covered by recent accumulations, or cut across by modern valleys of erosion, that without a careful survey no one could recognise that table-lands twenty miles distant were parts of the same ancient river deposit. In some cases the gold-bearing material has been covered by beds of basalt 150 ft. thick.* In many cases the erosion has been such that the bed of the old river is now 100 ft. or even 200 ft. above the surrounding country.

To work these deposits, careful surveys of the whole country must be made so as to be able to reach with a tunnel the lowest point of the bed-rock, which must be determined by sinking shafts upon it. The cost of this preliminary work may often be less than $100,000, and instances have been known where from want of proper judgment in the outset the whole of this sum has been lost. The location of the tunnel must be such that the pay dirt can be washed through it, and that it may form an outlet for all the material which is deposited after the

* A map of the basalt-capped deposits of the North Bloomfield Company is given in the Report of the U.S. Mining Commissioner for 1875, p. 116.

extraction of the gold. Its construction involves the building of miles of sluices to catch the gold and carry the dirt away; the damming of streams to save the winter's water supply; the storing up of billions of gallons, and conducting it in ditches, flumes, and wrought-iron pipes, sometimes forty, fifty, or even a hundred miles in length, the ditches alone costing in some cases from half a million to a million of dollars, and involving constructions which are marvels of lightness, strength, and engineering skill. The following table* gives a fair idea of the size and cost of the ditches in California:

—	Length. Miles.	Capacity in Miner's Inches.	Grade. Feet per Mile.	Size in Feet. Top.	Bottom.	Depth.	Cost.
							$
Smartsville ditches	...	5,000	9	8	5	4	1,000,000
Eureka Lake and Yuba ditches	163	5,800	...	...	...	...	723,342
N. Bloomfield ditches and reservoirs	157	3,200	12 to 16	8¾	5	3½	708,841
South Yuba ditches	123	7,000	3 to 13	6	...	4 to 5	
Milton ditches and reservoirs	80	3,000	12 to 15	6	4	3½	391,575
Spring Valley and Cherokee	52	2,000	...	5	...	3½	
Hendricks	40½	...	6 to 12	5	...	2	.136,150
Blue Tent	32	18,000	10	8	6	4	150,000
La Grange	20	27,000	7 to 8	9	6	4	500,000

This water is discharged through iron nozzles with a velocity of 150 ft. per second, and at the rate in some instances, of 4,220,000 cubic feet in twenty-four hours, against a bank from 250 ft. to 300 ft. high, and washes the earth into wooden sluices paved with rock or wood. To make the action of the water more effective, the bank is mined and fired, single blasts of from 1500 to 2000 kegs of powder being made.

As everything in the bank must come down, huge cranes with booms 90 ft. long worked by hurdy-gurdy water-wheels are set up to lift the boulders, undercurrents to catch the gold, grislies

* Burchard, "Production of Gold and Silver in the United States," p. 318. Washington, 1881.

to carry off the stones, drops to break the materials up. The sluice itself has to be paved with stone or wood and furnished with branches, so that one part may be repaired without stopping the work on the rest. Every part must work harmoniously with the other parts, and must be adapted to the lay of the ground, and every possible resource in the surroundings made use of for its successful working. Individual enterprise could do little or nothing with such claims, but the constant and large returns show that the immense outlay is fully justified.

The gold is caught in mercury, put into the sluices between the pavement and riffles. The greatest difficulty is not so much to catch the gold as to get rid of the tailings or material that has been treated. This involves the construction of miles of tail-sluices and the destruction of land and of streams by depositing on and in them stones and sand to great depths, but it saves for the use of the country the very large amounts of gold deposited in exceedingly small quantities in the ancient river-beds of California. No one who has not visited these mines can have any idea of the devastation produced by this washing away of hundreds of acres of surface and hundreds of feet in depth of the ground of these gold-bearing districts. It will be many years before this question of local land devastation will need to be considered, but the filling up of the rivers and streams is engaging attention now.

The cost of producing one Troy ounce of metal is given below:*

	La Grange Co,	No. Bloomfield Co.
	$	$
Water	1.43	2.09
Labour	6.85	3.93
Materials	1.81	0.88
Explosives	...	0.98
Blocks and lumber	...	0.50
General expenses	0.94	0.70
Contingent expenses	0.26	...
Taxes	0.09	...
Total	11.38	9.08

* The value of the metal was $18.53 per ounce.

The height of the bank washed down and the yield for several mines is given below:*

	Height of Bank in Feet.	Yield per Cubic Yard in Cents.
Smartsville Claims, Yuba Co. ...	112	19.5
Blue Tent, Nevada Co.	180	15
North Bloomfield, Nevada Co. ...	180 to 260	4 to 6.5
Gold Run, Placer Co.	200	4.8
Columbia Hill, Milton Co.	100	4.33
La Grange, Stanislaus Co.	18 to 100	2.5 to 15.5
Patricksville, Stanislaus Co... ...	40 to 60	4.33 to 18.5
Dardanelles, Placer Co.	150	13

The cost and yield per cubic yard of some of the mines is given below:

	Cost. $	Yield. $
Roach Hill	0.6	0.60
Richardson	0.03	0.15
Iowa Hill...	0.025	0.71
Independence	0.02	0.25
Wisconsin	0.02	0.125

The cowhide hose used at first soon became rotten and burst. This was succeeded by one made of heavy duck from 4 in. to 10 in. in diameter. This was made sometimes of one, sometimes of two thicknesses. Such a hose will bear a pressure of 50 ft., but no more. To make it stronger it was surrounded by iron rings 2 in. wide and 3 in. apart, which were held in position by four ropes distributed evenly in the diameter. Such a pipe was called a crinoline hose, and would support a head of from 150 ft. to 200 ft. of water. This was subsequently abandoned for the sheet-iron pipe now generally in use. The profits of this kind of mining do not depend so much on the yield of the pay dirt as they do upon the cost of the water, the expense of getting rid of the tails, and the facility of working which depends on the lay of the ground. A claim well situated can work a much poorer gravel with a profit than one less advantageously placed.

* Burchard, "Production of Gold and Silver in the United States," p. 318. Washington, 1881.

Old Mexican Rock Breaker.—Fig. 1.

TREATMENT OF GOLD QUARTZ.

As the placers grew poorer and the search for other sources of gold became active, the prospectors soon found gold in veins, and these were then explored and worked. It was usually found in a hard rock which was ascertained to be quartz, but when it was afterwards discovered in other rocks no attention was paid to the correct name of the stone. The gangue of the gold was always called "the quartz," no matter how hard or how soft the rock was, or what its chemical composition might be; the "quartz," if it was quartz, was said to be hard, if it was slate it was said to be soft. To "get" the vein rock required capital and a much higher degree of skill than had as yet been required for the working of the shallow placers. Deep quartz mining could not be carried on by individuals, and mining companies were formed to mine the quartz and separate the gold. At first these were all undoubtedly legitimate enterprises, but it was not long before some men found, or thought they found, a more expeditious road to wealth in mining shares than in mining quartz —a practice which very soon brought discredit on all kinds of enterprises in the Western States and Territories.

To get the gold out of the vein rock it had to be crushed. This was done, in the very early days before mining companies were known, with a large rock bound to a pole supported on a crutch so as to have a long purchase, Fig. 1. The rock was raised by one man and allowed to fall on the ore, while another kept the pieces of ore from flying away with a stick of wood. It was not long before this rude Mexican hand labour was replaced by the arastra, which was a hearth or bed of uncut stones arranged in a circle from 10 ft. to 20 ft. in diameter, with a curbing 2 ft. deep on the inside, over which large stones were dragged by a single mule. It was necessary to run this machine for at least a week, and sometimes for two or even three times as long, to make it worth while to clean up, Fig. 2. The joints between the stones were so open that the mercury and amalgam settled down between them so that the whole bed had to be dug up, the earth

carefully collected and washed, and the hearth replaced before a new charge could be made. The hearth was then improved by making it of cut stones with very close-fitting joints laid in cement. To the upright part one arm or two arms at right angles to the diameter of the bed were attached, and to each end of the arms stones weighing from 400 lb. to 500 lb. were fastened by chains so that the forward end was about 2 in. above the hearth while the other end dragged on it. One mule was counted for each stone, so that there were one or two-mule arastras. To make the charge for an arastra 10 ft. in diameter, 500 lb. of quartz broken by hand to the size of a pigeon's egg was put in, and the mules driven for four or five hours. Water is introduced to make a paste of the consistency of cream, and then quicksilver is added and the mule driven for two hours more. The paste is now thinned with water, the mule driven slowly for half an hour, the mud is run off, and another charge introduced. Four charges can be made in twenty-four hours, but two are generally all that are made, so that half a ton per day is about the limit of capacity of a 10-ft. arastra. The clean-ups are easily and frequently made, so that this machine is a much better one than the first, which was built for very rude work.

It is quite easy to see from this description where the idea of pan amalgamation originated. It needs but a few mechanical appliances to be added to have the description of the pan as now used. The arastra is still used in Mexico, and as it costs next to nothing to erect, it may be used as a prospecting tool. The principle on which it is constructed is an excellent one, the grinding perfect. It gives a large percentage of the assay value, but is too slow for use where large quantities are to be treated.

The Chilian mill, Fig. 3, was used about the same time, and consists of a circular bed like the arastra, but the ore is crushed by one or two large circular grinding wheels made either of stone or of cast iron, which roll around on their edges. The Chilian mill is often used to grind the ore previous to the treatment in the arastra. Used alone it is more expensive than the arastra, and does not do its work any better. It was not long before the eager and impatient miner found that the capacity of the arastra and of the Chilian mill was not sufficient for the profit which he

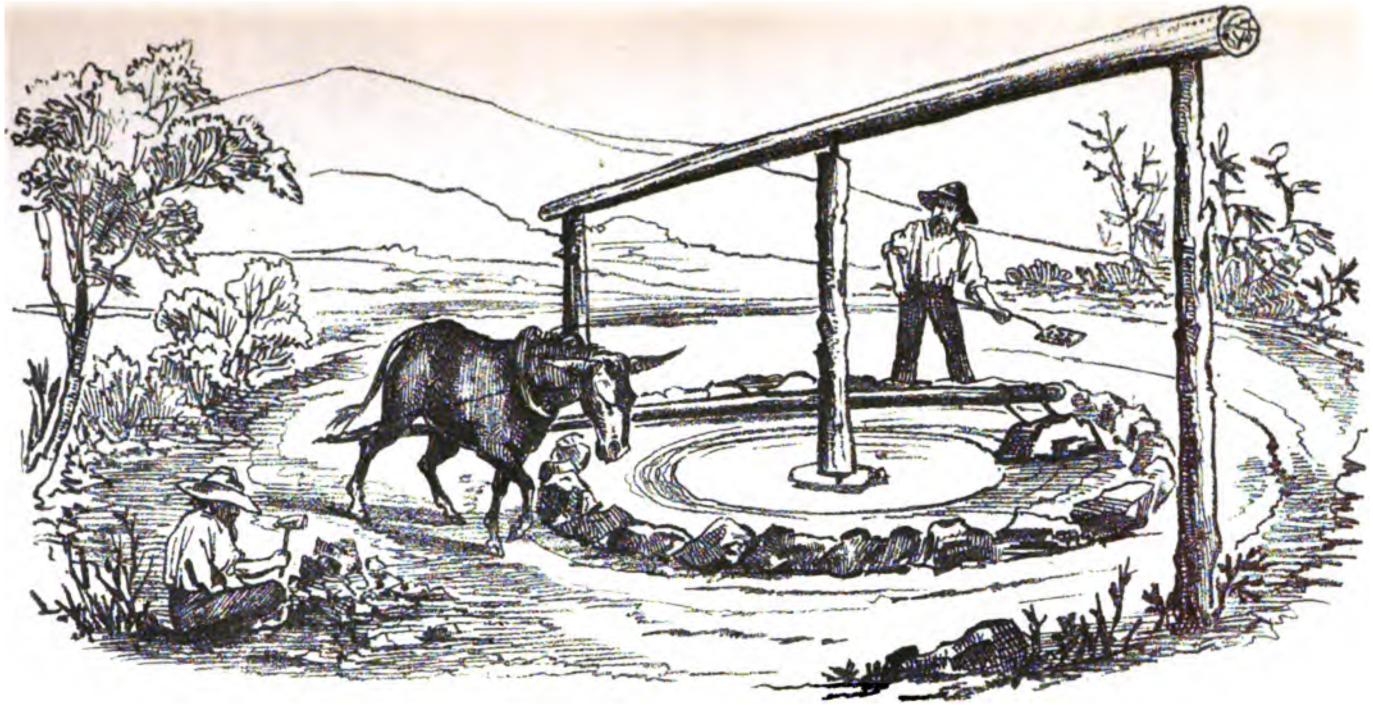

FIG. 2. MEXICAN ARRASTRA FOR GRINDING ORES.

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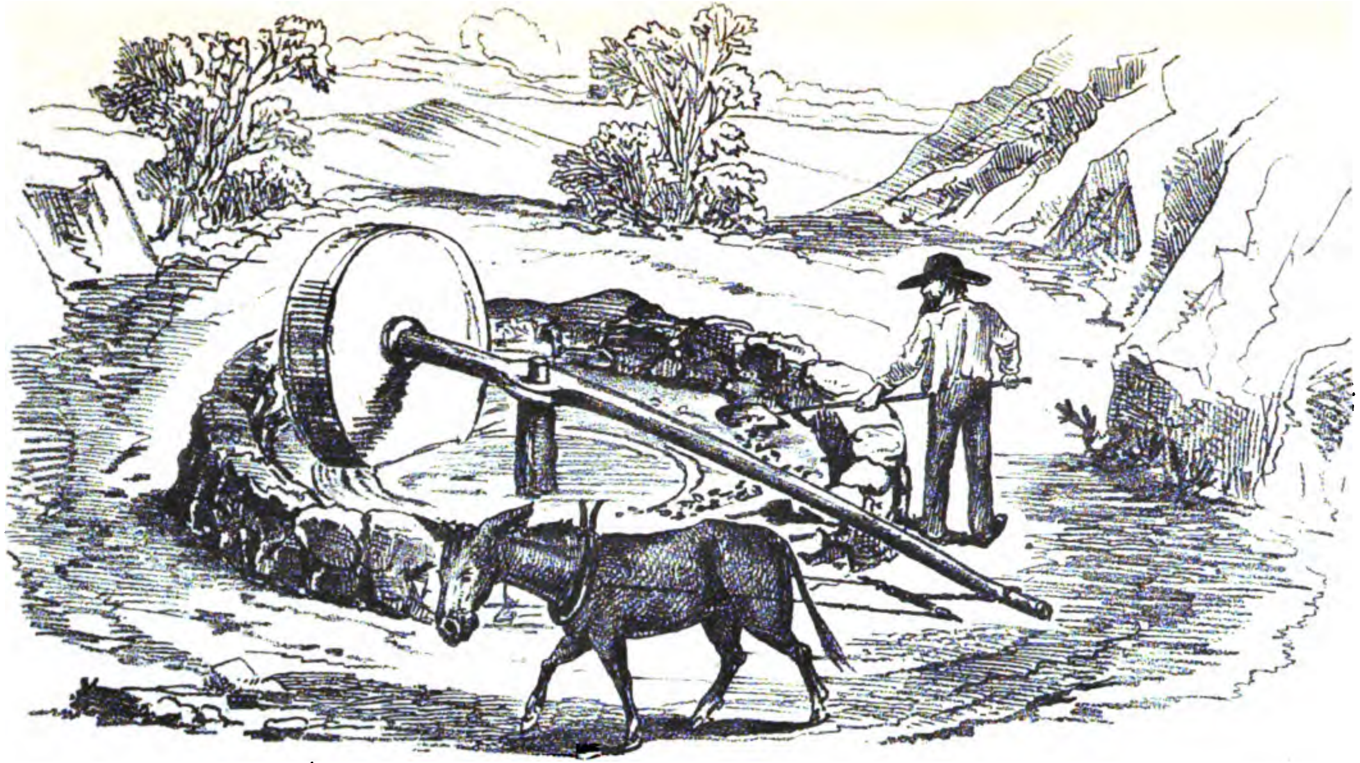

Fig. 3. Old Chilian Mill.

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wished to get out of his ore, and he looked to some of the crushing machines used in Europe to increase the capacity of his mill, and at the same time his hoped-for profit. The stamp-mill seemed the only one likely to be of use, and this was adopted. There were then but few mines whose output would justify the erection of stamps, but where it would do so they were built. In other cases the miners were obliged to depend either on the stamps of some neighbouring mine being sufficiently at leisure to do other work than their own, or to mills being erected independently of the mines to do whatever work was brought to them. To distinguish such mills from the others they were called "custom mills." They worked for everybody and at a fixed price, reserving the tails for themselves, which they usually took care should be rich.

The lumbering German stamp, with its wooden stems 8 in. square, its complicated and cumbersome cam shaft, and its inefficient mortar, was first built. Then some Cornishmen suggested the idea of the Cornish stamp, which, with its rectangular iron stems but inefficient mortar, was quickly put up, and was for a short time such an improvement that it was considered to be the *ultima thule* of crushing. This gradually grew into the rotating California stamp and its adjunct the Blake's crusher, with its increased capacity and much better mechanical appliances, which is now almost exclusively used in California. The stamp with its head weighs from 500 lb. to 1000 lb. Its capacity is, for very hard rock 1½ tons, and for very soft rock 4 tons, in twenty-four hours. The average will be about 2 tons. It is built in batteries of five stamps each, so that each battery may be counted for 10 tons in twenty-four hours—a capacity which, considering the cost, is very limited. It is in universal use in California, where no other means of crushing has been, until within a few months, able to compete with it. In other parts of the country it has grown into the Ball's stamp, a very large and heavy rotating stamp, in which the force of the blow is increased by the pressure of steam to such an extent that a single machine with one stamp-head is capable of crushing 110 tons of the hardest rock in twenty-four hours, and thus a single machine becomes equal in effectiveness

to a 60-stamp mill, which is the size of some of the largest mills in the West. No investigation has yet been made to ascertain the comparative efficiency of the use of power in these two machines. The Cornish, California, and Ball stamp are working side by side on Lake Superior, and it is to be hoped that experiments to settle the question of efficiency will be made. Since the leaching processes are becoming so generally introduced, great attention is being paid to the use of steel rolls. They have been used with very great success in a number of mills. Their small cost, great output, and settled efficiency in all cases where coarse crushing has been required for leaching purposes, has led to their use for fine crushing in which for purposes of amalgamation they seem to answer all the requirements. A well-constructed pair of 26 in. rolls is equal in efficiency to a 50-stamp mill, crushing through a 40 screen. There are several other crushing machines which have lately been introduced, but none of them have been fully tested, though some of them seem to be of great promise.

But it was not sufficient to crush the rock, the whole bulk of the crushed material had to pass over or through mercury in order to extract the gold and silver; and the quantity of this rock was so large that it produced losses in mercury so great as seriously to diminish the profits. Amalgamated plates and some free mercury were placed in the mortar, but these, unless the pieces of gold were large, could never catch more than 80 per cent. of it, and when the pieces were small not more than 70 per cent. If the gold is associated with silver as in some of the Nevada mines, but very little of it is caught in the mortar, and the amalgam produced is so light and spongy that it is liable to be carried off; so that where gold ores contain much silver, lead, or antimony, battery amalgamation is unadvisable. There are many gold-mills where it is never used, though some of the best conducted ones do use it. There must therefore be a sluice at the end of the splash-box to catch the fine particles coming from the mortar. The sluice is thus, as an appendage to the battery, as important as the battery itself. It serves a very different purpose from the placer-sluice, where the dirt is to be transported, the clay broken up, and the stones carried along so

as to allow the gold to sink where the mercury is. The battery-sluice has neither to deal with stones, dirt, nor clay, but only with crushed ore. It has to treat much less material; it is therefore much shallower and has a lower grade. When the ore contains copper and iron pyrites, these can by proper treatment be concentrated by it and saved. These battery-sluices are used with amalgamated copper plates with transverse riffles in which mercury is placed, and with the different kinds of blankets. Great attention is being given just now to iron riffles, not only for battery but for placer sluices. The experiments made give reason to hope for a great saving of the precious metals by their use wherever sluices are used. In some localities the battery-sluice is used in connection with such machines as the Attwood's amalgamator and the Eureka Rubber, but the tails are still rich. Some one suggested the use of cowhide with the hair on and the grain of the hair turned against the current, over which the tails were allowed to flow in order to catch the heavy materials, while the lighter ones were carried off by the stream; and out of this grew all the different styles of stationary and revolving blanket-sluices. Still the tails showed by assay that they were rich, and it was then found that the gold was contained in iron pyrites. It is very remarkable that at this period when Mexicans with their arastras and slow but extremely simple processes could make $50 to $60 a day, the best stamp-mills with the most efficient machinery, working on the same rock from the same vein could not recover from the same ore more than $15 to $20. In some instances, with the best modern machinery an ore yielding by assay $700 to $800 in gold did not yield more than $20 to $30 when stamped and treated with the usual modern appliances. The slower but more perfect process of the arastra had brought the pyrites into such a fine state of division, and had by constant abrasion of the stone rubbed it so bright, that the "quick" took it up, while the more rapid process of the stamp did not. As soon as it was thoroughly understood that the pyrites contained the gold, concentrators of different kinds, with or without buddles or keeves, were used to catch it, as no smelting process could be used in the localities where the pyrites was found, and the gold in the pyrites could not be separated with mercury.

The tails are therefore generally kept to be put through a series of concentrating machines whose object it is to catch the pyrites and possibly some amalgam, but which save none of the gold which has escaped as float. It is a question of grave importance how this gold may be caught, or, better, how to prevent its getting into the condition in which it cannot be caught. The miners call the gold that escapes the mercury, "float" and "rusty" gold. That some gold exists in the ore in such fine particles that it will float seems undoubted; against this there is no remedy. It is also true that the heavy stamp falling on the ore does make float-gold of some of the precious metal not in that state in the ore, but this is not the principal source of loss. If a piece of pure gold, which amalgamates readily, is pounded with a hammer on a clean smooth anvil, it is very soon put into the condition in which mercury will not touch it. I have had such a piece of gold in contact with mercury for more than a week without amalgamation. From this condition the gold can readily be recovered in the laboratory, but it is doubtful if it can be saved in the mill. Something must be done to avoid it, as there seems to be no doubt that some part of the gold escapes amalgamation from this cause. But there is gold which is really rusty, not covered with a coating of oxide of gold, but of something which prevents contact with the mercury. Absolute contact is necessary to amalgamation, and the thinnest film between the gold and mercury will prevent it. One of these coatings is oxide of iron, which does not occur very often and is very easily removed by abrasion, and another is said to be silica; grease from the stamp, or which may be in the water used, will also produce it. I have shown* also that a small amount of sulphuretted hydrogen or sulph-hydrate of ammonia will produce exactly the same effect, leaving an impalpable greasy film on the outside, which prevents the action of the mercury. In other words, a dirty mill, water which is not clean and not carefully protected from drainage, will affect the gold. No one has until now thought it necessary to call attention to this subject.

I have elsewhere shown† that there are many other interesting

* Trans. Inst. Min. Eng., vol. ix., p. 633.

† *Ibid.*

facts relating to the metallurgy of gold which have escaped observation. How far they may affect its extraction from the ores is yet to be seen. It seems, however, certain that we are creating some of the difficulties, and that some other machine than the stamp will have to be used in the near future for pulverising gold ores.

The cost of the treatment of a gold ore, including the mining, varies very much and depends on a great variety of circumstances, such as the hardness of the rock, cost of transportation, price of labour, &c. It may vary from $1 to $10 per ton. It is usually higher in custom mills than in those mills owned by the mine, where the quantity treated is very large. In such mills the cost of crushing, varying with the rock, will be from $1 to $2 per ton. In the custom mill where only a small quantity is run it may be as high as $5 for the same ore. The yield of the ore and the consequent profits are very variable. Ores yielding as low as $5 have been treated in a large way with a profit, but so much depends both on the nature of the ore and on local circumstances, such as the management of the mill, that it is quite impossible to say how poor an ore could be worked. Perhaps no one thing has been so great a stimulus toward the perfection of the gold processes of California as the discovery of the mercury mines there. The demand for a large quantity of mercury so stimulated the rival companies that they improved their process to cheapen their production; but after they had increased their output, they found themselves with such active competition that the price of quicksilver has fallen to less than one-fourth of what it was, so that the free use of mercury is no longer feared in the poor mines, and the ominous question of the loss in mercury does not loom so high nor figure for so large a part of the cost in the processes as formerly.

For a number of years all the metallurgical losses of gold mining were attributed to sulphurets, and processes for working these were invented without number, most of which have died out long ago. The beautiful process of Plattner for roasting these sulphurets and then extracting the gold by chlorine was adopted and improved upon, and for a long time seemed to have solved the question, but little by little it was ascertained that

there were certain substances contained in the gangue of the rock, such as lime and magnesia, and certain other substances which might be associated with the gold, such as lead and zinc, which would be attacked by the chlorine, and that there were circumstances in which, after the gold was in a soluble form in the tanks, it might be again precipitated in the insoluble material of the tails, and thus be lost. Recent experiments have been made with nascent chlorine, with this gas under pressure, and other modifications which gave promises for the future which have not been fulfilled. It is apparent, however, that Plattner's process in any or all of its modifications does not cure all the evils, because it does not cover every variety of case, but is only applicable to certain ores in which there is nothing but the gold in the ore which would be attacked by the chlorine, and nothing which would prevent its acting on the gold.*

If the ore contained any silver this would be attacked, and a coating of insoluble chloride of silver would be formed over the gold; this would prevent further attack by the gas, and not only would the silver be entirely lost, but any particles of gold contained in it would also be lost. A very careful dressing might separate some of the gangue attacked by the chlorine, but it could never separate the whole, and any part of it remaining would be a source of loss; so that as a general rule it may be stated that when the ore contains anything but gold which the chlorine will attack, the process is not applicable. Besides, Plattner's process depends upon delicate chemical reactions. If, for instance, any of the sulphate of iron resulting from the roasting is left in the ore, as soon as the gold has been put into a soluble condition and is leached with water, a part of the gold dissolved is thrown down by the sulphate of iron. Certain organic compounds produce the same results. The gold thus previously rendered soluble is then precipitated in the gravel of the filter on the bottom of the tub and is lost. I saw such an accident at Grass Valley, and on examining the tails on the

* The carbonates of lime and magnesia are attacked by the chlorine, and use up the reagent without producing any effect. Clay becomes diluted in the water. The earthy particles in suspension settle on the small flakes of gold and prevent the action of the chlorine.

filter found them very rich in gold. It is not very wonderful, therefore, that a process which demands such nice working and depends on exact chemical reactions does not succeed very well where there are no trained metallurgists. I have known an expensive plant abandoned and the process brought into great disrepute for one of the reasons given above, when if there had been a trained metallurgist in charge, there is every probability that the defect would have been remedied and the process would have succeeded.

It sometimes happens, too, that where the process is otherwise applicable, certain substances held in solution in the water of the district, if the gold already dissolved in the water is allowed to remain for a short time in the tanks, and sometimes by simply filtering through them, cause the gold already dissolved to be precipitated on the filter and thus to be lost. In some cases where the works might have been successful had there been only one, there were so many competing for the product of the district, that they were obliged to lie idle more than half the time or pay a higher price for the concentrates than they were worth, in order to keep at work.

As a result of the knowledge that gold could be recovered from the tails, there followed a series of concentrating machines, of blanket-sluices, of different kinds of amalgamators, of pans for grinding and amalgamating, and of single machines for doing a dozen things which require to be done each by a separate machine in order to be well done.

For a number of years the idea that gold and silver could be concentrated by means of smelting had not occurred to the people in the West. The miner had divided his ores into placer and milling ores, and the latter into free milling and rebellious, by which he meant ores which would or would not readily amalgamate. Such rebellious ores as would not yield to mercury after roasting, or to which some leaching process was not applicable, were treated for what could be got out of them. To make it possible to treat them by any other process to recover the gold, either copper or lead must be in such quantities that smelting will be remunerative. Such is usually the case to make the treatment for gold alone possible. It occurs in Colorado,

where a complicated European process has been introduced with such modifications as were made necessary by local circumstances to treat ores of copper or lead containing gold, to which all the rebellious ores of the district can be profitably added; but the lead is always lost, while the copper is saved as a by-product. The gold thus concentrated in Colorado in a copper product is afterwards separated with little loss. If the gold is concentrated in a lead product, silver must be present, when the gold follows the silver and is afterwards separated from it.

Such methods of smelting are not common. Gold is not usually found in the United States in paying quantities in copper or lead ores, nor have those ores as a general rule been found in any very large quantities in the districts where rebellious gold ores occur, nor if they did, could the separation be undertaken except by men of great skill both in metallurgy and in financiering, as the failure of enterprises of this kind based on either the one or the other, but without both, has shown.

That gold associated with copper, lead, silver, and zinc, could be separated by smelting, when either lead or copper ore could be had in sufficient quantity, was learned after a long trial, but not until hundreds of thousands of tons of rich tailings from amalgamating processes, containing over four pounds of mercury to the ton in addition to large quantities of gold and silver, had been allowed to run to waste in the streams which flow down the canons of the mining districts.

We cannot say that we have reached the utmost limit of metallurgical progress in the economical separation of the precious metals, for it has been ascertained that in the hydraulic mines from which by far the larger part of the gold of California is produced, notwithstanding the great cost of their plant, not more than 33 per cent. of the total amount of gold contained is saved.

The observer who takes the pains to calculate the millions of dollars contained in the tails which have been allowed to flow away looks aghast at these statistics, but the miner, when he comes to consider them, makes the very intelligent reply that while he gets out only 33 per cent. he makes a profit for himself

and his stockholders, but when he endeavours to save any portion of the other two-thirds he makes either less profit or none at all, according to the amount of increase which he attempts to make. It is a matter of great regret to him that he cannot save the 67 per cent., but his regret is not that it is wealth lost which would be added to the resources of the State, but that he cannot have it at the bottom of his own pocket ; so he abandons it philosophically to the future miner with the expectation that nature will some time concentrate these tailings, or that some process will be invented by which the vast value contained in the tails that are now lying piled up in many of the cañons of the West can be utilised. In the mean time the filling up of the river-beds and the covering of some of the agricultural lands have become a serious political question, and it may be that in the near future California, which would probably not have been settled but by a mining population, will have to decide whether it will allow the agriculturist, who does not add nearly so much to the wealth of the State to drive the miner out of the deep placer districts where he settled long before the farmer came there, since the granger element threaten proscription in the shape of legislation which compels the miner to retain his tailings in the side cañons and not to allow them to escape into the rivers or over the arable lands. This questior has recently come into the courts of California by an injunction restraining the Miocene Mining Company of Butte Co. in that State, who own 1500 acres on Feather River of placers 30 ft. deep, from discharging their tails into this river. The magnitude of the interests involved is shown by the following figures taken from the San Franciso *Alta* of June, 1881 :

Miles of ditches and canals in the counties affected...	6,000
Number of men employed, from	5000 to 10,000
,, Chinamen	500
Capital invested in the mines	$150,000,000
Yearly product	$12,000,000 to $15,000,000
Mining population affected by the suit	130,000
Agricultural	60,000

It is estimated that should this injunction be made permanent,*

* The injunction, after appeal, was made permanent by the Supreme Court of California in November, 1884.

the amount of farming land now under cultivation which will be rendered valueless by the removal of the mining population will be equal in money value to the total amount of the value of the mining property involved. The State will probably then learn too late that a valuable industry has been destroyed by people who found the mines working when they settled on the lands they occupy, and must have known beforehand what was in store for them in the near future. The industry so destroyed it will be difficult to re-establish, and it is very doubtful whether the State will be gainer by it. The immediate effect of the action of the courts in restraining the hydraulic mines from depositing the tails in the usual manner, is seen by the diminishing gold production of the State as shown on page 35.

It is quite as impossible to form any adequate idea of the amount of gold produced during the first periods of mining as it is to estimate the losses. No attempt was made to make any record at the time. The records of the custom-houses are the only ones that were made, and are necessarily imperfect; they show that between 1848 and 1854 over $350,000,000 had been recorded, but previous to that time very large sums had been carried away by those going either to Europe or to the East, either in their effects or on their persons, and while the amount carried by an individual was small, the aggregate must have been very large. In the next decade more perfect records were made, which show a gradual annual decrease, the highest point having been reached in 1853, when the amount recorded is $65,000,000, which is the largest annual production ever known in the State of Califonia, and the lowest $26,000,000 in 1864, the total amount for the decade being $450,000,000. The next decade to 1874 shows a product of only $220,000,000, the lowest amount reached being $17,000,000 in 1873. This rapid decrease gives a very fair idea of the amount which must have been wasted within the first seven years of the most prosperous time, when the records, though very imperfect, give an average of $50,000,000 a year, while the year 1873, when the records were most perfect, gave only $17,000,000. From 1873 to the end of 1882 the total product of the State of California was $170,113,000; the lowest annual product being $15,000,000 in 1877, and the highest

$18,000,000 in 1874. The following Table, showing the amount of gold produced in California in each year,* as compared to the total production in all the other States, has been prepared for me by the Director of the Mint at Washington.

Year.	Gold in Dollars.		
	California.	Other States.	Total Gold.
	$	$	$
1848 . . .	10,000,000	...	10,000,000
1849 . . .	40,000,000	...	40,000,000
1850 . . .	50,000,000	...	50,000,000
1851 . . .	55,000,000	...	55,000,000
1852 . . .	60,000,000	...	60,000,000
1853 . . .	65,000,000	...	65,000,000
1854 . . .	60,000,000	...	60,000,000
1855 . . .	55,000,000	...	55,000,000
1856 . . .	55,000,000	...	55,000,000
1857 . . .	55,000,000	...	55,000,000
1858 . . .	50,000,000	...	50,070,000
1859 . . .	50,000,090	...	50,000,000
1860 . . .	45,000,000	1,000,000	46,000,000
1861 . . .	40,000,000	3,000,000	43,000,000
1862 . . .	34,700,000	4,500,000	39,200,000
1863 . . .	30,000,000	10,000,000	40,000,000
1864 . . .	26,000,000	19,500,000	46,100,000
1865 . . .	28,500,000	24,725,000	53,225,000
1866 . . .	25,500,000	28,000,000	53,500,000
1867 . . .	25,000,000	26,725,000	51,725,000
1868 . . .	22,000,000	26,000,000	48,000,000
1869 . . .	22,500,000	27,000,000	49,500,000
1870 . . .	25,000,000	25,000,000	50,000,000
1871 . . .	20,000,000	23,500,000	43,500,000
1872 . . .	19,000,000	17,000,000	36,000,000
1873 . . .	17,000,000	19,000,000	36,000,000
1874 . . .	18,000,000	15,490,902	33,490,902
1875 . . .	17,000,000	16,467,856	33,467,856
1876 . . .	17,753,000	22,176,166	39,929,166
1877 . . .	15,000,000	31,897,390	46,897,390
1878 . . .	15,260,000	35,946,360	51,206,360
1879 . . .	17,600,000	21,299,858	38,899,858
1880 . . .	17,500,000	18,500,000	36,000,000
1881 . . .	18,200,000	13,500,000	34,700,000
1882 . . .	16,800,000	15,700,000	32,500,000
1883 . . .	14,120,000	15,500,000	30,000,000
1884 . . .	13,600,000	17,200,000	30,800,000

It is remarkable that while up to 1860 there was almost no gold produced in the States east of California, and that while in that year only $1,000,000 is reported, between 1860 and 1870 the

* The fiscal year ends the 30th of June.

amount produced gradually increased, until in 1866 it exceeded it, and only fell so as to equal it in 1870. In the next decade, from 1870 to 1880, the largest amount produced in all the other States except California was $35,946,360 in 1878, and the lowest $15,490,902 in 1874; with the exception of 1874 and 1875 the Eastern States produced more than California, and in 1878 more than double the amount produced there. The total yield of gold in the United States in the year 1884 was only $30,800,000, or less than that produced by California alone up to 1862, with the single exception of the year 1848.

The following Table, taken from the annual reports of the director of the United States Mint at Washington, gives the annual production of each gold-producing State from 1877 to 1884. It shows that while Nevada produced more gold in the years 1877 and 1878 than California, this latter State is still the great gold-producing State of the Union. The large quantity of gold produced in Nevada during those years is owing to the large proportion of gold contained in the silver of the ores of the Comstock lode. This remarkable lode, which is mined to a depth of over 3000 ft. deep, and contains about 185 miles of drifts, produced, in the twenty-one years ending June 30th, 1880, a little more than $306,000,000 of bullion, of which $132,000,000 was gold, and the rest was silver. Since the decline in the production of these mines, California has resumed her position as the great gold-producing State of the United States.

It is extremely difficult to ascertain the amount of gold contained in the world, and it may be said that the best estimates from the most reliable authorities are but the merest approximations. It is supposed, however, that the entire amount of gold in existence at the present time is not over $7,000,000,000. As the value of a cubic yard of gold is $9,000,000, if all this gold were melted into one mass it would contain about 700 cubic yards, which would make a block about 25 ft. by 25 ft. by 31 ft., or less than the cubical contents of an ordinary small city house.

VALUE OF GOLD PRODUCED IN THE UNITED STATES.

STATES.	1876.	1877.	1878.	1879.	1880.	1881.	1882.	1883.	1884.
	$	$	$	$	$	$	$	$	$
Alaska	...	...	...	6,000	7,000	15,000	150,000	300,000	200,000
Arizona	300,000	500,000	800,000	400,000	770,000	1,060,000	1,065,000	950,000	930,000
California	15,000,000	15,260,679	17,600,000	17,500,000	19,000,000	18,200,000	16,800,000	14,120,000	13,600,000
Colorado	3,000,000	3,366,404	3,225,000	3,200,000	3,400,000	3,300,000	3,360,000	4,100,000	4,250,000
Dakota	2,000,000	3,000,000	2,420,000	3,600,000	4,500,000	4,000,000	3,300,000	3,200,000	3,300,000
Georgia	100,000	100,000	90,000	120,000	150,000	125,000	250,000	199,000	137,000
Idaho	1,500,000	1,150,000	1,200,000	1,980,000	1,930,000	1,700,000	1,500,000	1,400,000	1,250,000
Montana	3,200,000	2,260,511	2,500,000	2,400,000	2,500,000	2,330,000	2,550,000	1,800,000	2,170,000
Nevada	18,000,000	19,546,513	9,000,000	4,800,000	2,700,000	2,250,000	2,000,000	2,520,000	3,500,000
New Mexico	175,000	175,000	125,000	130,000	120,000	185,000	150,000	280,000	300,000
North Carolina	100,000	150,000	90,000	95,000	75,000	115,000	190,000	167,000	157,000
Oregon	1,000,000	1,000,000	1,150,000	1,090,000	1,000,000	1,100,000	830,000	660,000	660,000
South Carolina	...	...	...	15,000	18,000	35,000	25,000	56,500	57,000
Tennessee	...	...	...	...	2,000	5,000	...	...	...
Utah	350,000	392,000	575,000	210,000	200,000	145,000	190,000	140,000	120,000
Virginia	50,000	...	...	10,000	11,000	10,000	15,000	6,000	2,000
Washington T.	300,000	300,000	75,000	410,000	100,000	120,000	120,000	80,000	85,000
Wyoming	...	...	...	20,000	7,000	5,000	5,000	4,000	6,000
Other sources	25,000	25,000	50,000	14,000	10,000	...	...	17,500	76,000

TREATMENT OF SILVER ORES.

As gold grew scarcer, silver ores were looked for and became an object of great interest. At first only the rich outcrops of the free-milling lodes were worked. The ores from them were treated by the old Patio processes, which are still used in Mexico, and are characterised by the use of arastras and Chilian mills. Occasionally the Caso method was adopted by some persons who had seen it at work or had heard of its working in Chili. The Caso method, working quicker than the Patio, was adopted in some places; the bottom of the box was replaced by iron, and then the sides, and then the idea of grinding suggested itself, until the amalgamation pan in all its varieties grew up little by little. It was at first thought that the pan could be used equally well for both grinding and amalgamating, and some persons still use it for both purposes. But it has been pretty well settled of late years that the machine can do only one kind of work well, and that any other kind of work forced from it is done at the expense of the yield.

The Freiberg barrel did not meet with much favour in the West. Most of the emigrants of 1848 were of Anglo-Saxon origin, and when the German came with his slow but certain methods, a few mills adopted the barrel. But in the early days quick returns were demanded. The pan was an American invention. It, moreover, had grown on "the coast,"* and every one knew, or thought he knew, how to use it. Every machine shop was ready to make it, and was more or less interested in one of the many patents taken out for a new one. It gave a return in from three to five hours, while the barrel worked twenty-four before the amalgam was extracted. The pan required nothing but the ore, mercury, "the chemicals," and water. These had to be used in the barrels, and in addition some iron. Balls were used at first, as in Europe, then pieces cut fron stamp stems, and then mule and horse shoes, and finally anything made

* All the Sierra Nevada district is called "the coast" in the West.

of iron whether adapted by its shape to the purpose or not. The result was imperfect, partly from the want of material adapted to the purpose, and partly because it was not always possible to get the material wanted in a given time. The consequence was that most of the mills that put up barrels found no one to run them, or were obliged to keep one set of men to run and another to repair them. Yankee wit did improve them so as to make each barrel independent by using friction clutches, and by doing away with the cogging by the substitution of friction gear; but a change in the administration usually threw the barrels out and put in the pans, so that the barrel, while everywhere condemned in favour of the pan, has really not been fairly tested. It is cheaper to erect and cheaper to run, but slower in action, and, with friction gear, ought to compete with the pan.

Amalgamation, whether in the barrel or the pan, was first used only for ores that are now called free-milling—that is, which will amalgamate immediately with the mercury. But it became evident soon that there was a large amount of ore which would not amalgamate. It contained sulphur and other substances which either prevented the action of the mercury altogether or caused a great loss of it. Such ores were called rebellious, and at first were not used; afterwards they were roasted, the principal object of the roasting being not only to drive off the sulphur which was present, but, by adding a little salt, to convert the silver into chlorides which could be easily attacked by the mercury, which process was called the Reese River Process, in distinction from the Washoe, which is the free-milling.

Up to this time stamping had generally been done wet in order to avoid the losses occasioned by the dust, as there was no reason why, in the Washoe process, the wet ore should not be delivered to the pans as soon as it settled sufficiently to form the "pulp," but, with rebellious ores in regions where fuel was generally scarce, too much heat would have to be used in driving out the moisture from the ore, and dry-stamping took the place of the wet. As the ore came from the mine damp, drying floors, made by running the flues of the roasting furnace backwards and forwards under the iron plates covering the floors behind the stamps, were used, and the ore from the mine was spread out

over these plates until it was dry. As there was nothing but the force of the blow from the stamp-head to deliver the ore from the screens, the operation was slower, and as no water current carried the ore away from the front of the stamps, endless chains were placed in the dust-tight boxes which enclosed the front of the mortar-screens, which delivered the crushed ore into bins in the roof of the works, whence it was delivered, by spouts, into the furnace.

Every kind of furnace for roasting was invented and tried. These were usually some kind of reverberatory furnace, and were the subject of a large number of patents, and were altered and modified with more or less permanent success. Attempts were made to make the work wholly mechanical by the use of revolving cylinders into which the ore and salt were charged by machinery, in order to get rid of the difficult hand labour required, and the necessary exposure to fumes in roasting in ordinary reverberatory furnaces. A few of these survive in the Bruckner furnace and its derivatives, but as a general thing the cost of repairs to these contrivances, and the necessity of more engine power or of separate engines in the absence of machine-shops near at hand, increased the expense of working beyond the gain in diminished labour.

On account of the large loss both in mercury and in silver which are carried off in the tails, mills called tail-mills have been erected, where, notwithstanding the large loss in mercury which is known to exist in the treatment, there is a constant increase in the quantity of mercury, and very often a large yield in silver. These mills have no work to do unless the tails are accumulated by damming the valleys through which the tail-streams pass. When the accumulated tails have been treated, the mills have no further use, and very often the heavy freshets do the work first by sweeping the tails away. Sluices of great length and width have been put up, but their tails are still rich. Many machines have been invented for catching mercury and amalgam by making the tails pass over revolving blankets, rubbers, revolving amalgamated plates, and many other contrivances, but they are not as yet successful in the commercial sense.

When lead, copper, or zinc were present in the ores in any

considerable quantity they became so rebellious that amalgamation was out of the question, and smelting, with its necessary adjunct of concentration, became necessary. As the dressed ores were rich and contained a large product condensed into a small one, and as this product was usually sold, sampling works sprung up, in which the value of a large quantity of ore was carefully ascertained by processes more or less mechanical, in which, as rapidity of execution as well as correctness of results were necessary, a number of tools for reducing the ore to powder now generally used were invented.

The first attempts at smelting, as they were usually conducted by persons of no great experience, were not very successful. In fact, the early history of the now very successful American methods is a record of failures. When the smelting of silver ores became a necessity, English methods were first introduced by the Cornish miners, only a few of the German and Swedish furnaces being used. But as the English type of furnace requires a considerable amount of good fuel, of a kind not generally found in the West, and the use of wood in them requires great skill, shaft furnaces gradually took their place, for the most part, for treating ores containing gold, copper, silver, and lead by smelting. Some of the processes adapted from the old works in Europe found themselves in circumstances where the conditions of transportation, labour, or fuel were such that they could not compete with other districts, so that they gradually disappeared, and were succeeded by the same processes in a new dress, or in a new phase, to such an extent that the plant and the process as it is now used in the West would hardly be recognised by their inventors. Little by little it was ascertained that when the ore contained any volatile material, although it might be in small quantities, it would carry off with it very considerable portions of the precious metal; and then arose the idea of condensing chambers, until gradually without any one person having invented them, the methods have grown into the simple and very beautiful processes which are now in use in the West.

At first the only fuel used was charcoal, and this, as wood was scarce, was sometimes made from dead or from float wood,

or from woods too light for the purpose, so that it very often happened that the charcoal would crush by its own weight, and would not stand a charge in the furnace at all. It then became evident that the ores must have a comparatively high yield, and that as they usually had a gangue composed for the most part of silica, coke was necessary, and this, with from 10 to 20 per cent of ash,* was imported from the East or from Europe at an expense in some of the works of $40, or even $60, a ton. As it became evident that coke was the necessary fuel, it also became apparent that something must be done to reduce its cost, and also the cost of the refractory materials. Water-back furnaces were then introduced, which consisted of cast or sheet-iron boxes rivetted together, surrounding the hottest part of the whole of the furnace and cooled with water. These furnaces were for the most part open-breast furnaces with front hearths, and were continually getting out of order from the formation of sows and bears which occasionally stuck to the bottom or formed engorgements in the furnace higher up. To avoid these the Arent's tap was invented, which keeps a very considerable quantity of lead on the hearth of the furnace at all times, and allows of the casting being done from the outside of the furnace, without interfering with the interior. In other works where copper is in large quantity, the ores are smelted for copper, and the silver and gold concentrated in one of the products, from which it is separated in the wet way by the skilful adaptation of old processes.

It now became evident that there were considerable mechanical losses from the metal being carried up and out of the chimney, so that in some instances in Utah as much as 10 or even 15 or 20 per cent. was carried off in this way. Attempts were then made to collect this material, as is done in Europe, in condensation chambers of large size and extent, and several systems of doing it have been invented. The simplest, the cheapest, and the most recent of these is used at Mansfield Valley, near Pittsburgh, Pennsylvania, and is an adaptation of the flue leading to the chimney by

* The high percentage of ash in the coke has in several instances caused the failure of works which, with a suitable fuel, might have been successful.

dividing it in two sections vertically. In the lower one of these, partitions 18 in. high, placed 4 ft. or 5 ft. apart, and one-third of the height of the flue, catch the dust by gravity, and as there is no velocity below, it remains there. The gas circulates above.

Generally the silver and the gold in a district where lead ores can be had, are concentrated in a pig lead improperly called "base bullion." In some few cases in the early days the German method of cupellation was used, but as this requires a maximum consumption of fuel, great skill, and a market for litharge, it was quickly superseded by the English method, which requires less skill, makes no litharge for sale, but required the poor lead to be concentrated into a rich one and that treated for gold and silver. The Patterson process by crystallisation for enriching the lead previous to cupellation was never extensively used here, principally because at the time when there was a large amount of work to be done the process had already been superseded. The lead directly from the furnace is now enriched by zinc desilverisation, and the rich lead cupelled in an English furnace.

The history of this process is very peculiar. Invented in 1842 by Karsten, it was declared a failure after a prolonged investigation by that very able metallurgist. It was reinvented by Crooks in 1858 in England, where it was not very successful, and was brought to this country as an English process. It was tried again at Tarnowitz, in Silesia, and was more or less of a failure there, and was then reintroduced into this country, and so many improvements made in it that to-day the American modification of it has become the perfection of a process, and the furnace used, a type furnace.

In order to use the method of desilverisation by zinc it is necessary that both the lead and the zinc should be very pure. To purify the lead small refining furnaces were used in Germany, containing two to three, and subsequently from five to six tons each. But in this country one of the first improvements made was the softening or purification of the lead in a furnace containing from 15 to 20, and subsequently as high as 26 tons; but as the hearth of such a furnace was difficult of construction, it was simply made in a cast or wrought-iron pan. This softened lead had to be discharged from the furnace, which was not an easy

matter, and the late Mr. Steitz invented a siphon to do it, which seemed to be the perfection of an instrument for this purpose.

The refined lead is stirred with zinc, the zinc-scums carrying the silver with them are liquated to separate the excess of lead, and the result is a very rich zinc alloy, containing a large amount of lead, which is granulated and distilled in retorts. The distillation in retorts promised at one time to wreck the process, as it had to be effected in small furnaces surrounded by coke, and the number of retorts broken was large, notwithstanding the use of Steitz's siphon. Petroleum was then tried with great success, lessening the breakage of the retorts due to the charging of the fuel and the poking of the fire. Subsequently Mr. Faber du Faur invented his tilting furnace, which allows of pouring the rich silver lead out of the retort without disturbing it, thus removing all the difficulty. The silver lead from which the zinc has been distilled is cupelled in an English furnace and cast into pigs. The lead from which the silver has been removed is refined in a furnace similar to the softening furnace, called a calciner. All the lead so refined is of the highest quality, fit for the manufacture of white lead. It is produced almost as a by-product, and at a low cost.

The improvements in cupellation have been, first, the invention of the water-back cupel, by the late Mr. Steitz, of St. Louis, upon which the lead could be brought up to fine silver, and the later invention of Mr. Eurich, of the Pennsylvania Lead Company, of going from the lead riches to silver 996 fine, on a hearth made of Portland cement, and casting directly from the cupel into silver bricks, by a simple arrangement for tipping the cupel.

It sometimes happens that silver can be extracted from its ore in the wet way. There are three principal methods which have been used for this purpose. The first was introduced in 1849 by a German named Augustine, and consists in transforming the ore into chloride by roasting the ore to drive off the sulphur and other impurities, grinding the roasted ore and then roasting with salt to form chlorides; then dissolving out the chlorides with a saturated solution of salt, precipitating the silver with copper,

compressing and melting the silver. It is usual to concentrate the silver into copper mattes for this process. Shortly after the invention of this process, another, much simpler, was invented by Ziervogel, which consists in roasting mattes to produce sulphates, decomposing all these sulphates except that of silver, and then dissolving out the sulphate of silver with hot water. Simple as it appears, this process is exceedingly difficult to execute, for it requires a very high degree of skill to seize the exact moment when all the sulphates of the other metals are decomposed and none of the silver is. If the sulphates are not all decomposed the silver is precipitated by them; if they are, there is danger that the silver sulphate also will be decomposed, and it will then be lost, as the oxide is not soluble. The practice, therefore, is to leave a little of the sulphates of these metals undecomposed, as the loss in this case can be calculated beforehand, while no one can tell what it will be in the case of too much roasting. As these residues are always rich, they are often treated by the Augustine process, the two being very advantageously used together in this country.

In looking at the silver process as a whole and comparing the cost, we find that the relations between the relative cost and quantity of silver extracted were very interesting.

	Relative Cost.	Relative Loss.
Amalgamation	2.2	2.0
Augustine process	1.8	2.0
Ziervogel „	1.0	1.0

The Ziervogel process is, both as to cost and residues, twice as advantageous as the others.

Still another process was invented in 1858 by an Austrian of the name of Von Patera, which consists in roasting, as in the Augustine process, leaching with hot water, before roasting with salt, in order to dissolve out any soluble salts, roasting with salt, then dissolving the chlorides with hyposulphite of soda, precipitating the silver with polysulphite of sodium, and then reducing the sulphide of silver. This process is easily carried out, in that the reagents can readily be had, and that none of them are wasted; but both the lixiviation and the precipitation require

such nice distinctions and such an exact chemical knowledge that it has not been very successful.

Early in the year 1884 this process was so modified by Mr. E. H. Russell, that poor ores with considerable quantities of base metals can be more economically treated than by the original process, and is applicable to many ores which heretofore could not be treated by any process.

The bullion which is produced as the result of treatment of any of the ores usually contains some small quantity of the base metals besides the gold and silver. The gold from California generally contains about 12 per cent. of silver, that from Australia 4 per cent. to 6 per cent. The amount varies from 3 per cent. to 25 per cent. The silver bullion often contains gold, as in the case of the Comstock, where one-third of its value is gold; and these metals must be separated in order that they may be alloyed to their proper standards for commercial uses. Neither pure gold nor pure silver is of any use commercially except for electro-plating; for all other purposes they would be much too soft. The process of separation is called parting. To effect this an alloy is made by melting, which usually contains three parts of silver to one of gold. In a single instance in California this alloy is three of silver to two of gold. The formation of this alloy is called inquartation. It is granulated and subjected to one of three different processes; the silver is dissolved out by either nitric or sulphuric acid, and in both cases the residue not dissolved will be gold. The nitrate of silver siphoned off from this is diluted with water and precipitated with salt. The chloride of silver so formed is reduced to a metallic state with sulphuric acid and zinc, and the silver melted into bars whose fineness is stamped on them, and they are then used for commercial purposes. In the case of sulphuric acid, sulphate of silver is formed, which is diluted with hot water and precipitated as metallic silver by copper. The spongy silver is pressed into cakes by a hydraulic press and melted into bars. The gold is collected, melted, and run into bars. The nitric acid method has been generally abandoned, because it poisons the neighbourhood with fumes. The sulphuric acid process, which is a little cheaper, has taken its place, except in California, where

a very beautiful method, invented by Mr. Gutzkow, has taken its place. This method, which is very ingenious, much quicker, and gives better results than the others, was introduced in San Francisco in 1867. Most of the alloys are not granulated; they are inquartated and dissolved in sulphuric acid, in bars. The sulphate of silver is crystallised and a solution of sulphate of protoxide of iron run through it, which reduces the silver to a metallic state; the iron solution becomes sulphate of sesquioxide of iron, and is restored to its original condition with fresh iron and used again. The silver is pressed and melted as before, and the gold from the pots treated in the same way. This process is not only very simple, but is in a chemical way one of the most beautiful known.

The amount of silver produced in the United States previous to 1858 was so insignificant that no statistics have been recorded. In that year it was only $500,000; in 1859 it was only $100,000; in 1860 it was $150,000; but in the following year, 1861, the amount of silver began gradually to increase, until the year 1870, when it was $16,000,000. The total amount produced in this decade from 1860 to 1870 was $84,300,000, the lowest amount being $150,000, in 1860. From 1870 to 1880 the amount of silver becomes a very considerable factor in the world's production of this metal, the highest amount being a little over $45,000,000, in 1878, and the lowest $16,000,000, in 1870, the total production for the decade being $374,922,260. If these amounts of silver are added to the amounts of gold shown in the table, page 49, it will be seen that for the decade from 1860 to 1870, the highest production of precious metals was in 1869, $61,500,000, and the lowest was $43,700,000 in 1862, the total being $555,150,000. For the next decade, from 1870 to 1880, the highest production was in the year 1878, $96,487,745, and the lowest $66,000,000, in 1878. The largest amount of silver produced by any one State during the year 1883, was $17,370,000, obtained in Colorado, and the next largest $6,000,000 from Montana. In the year 1877, Nevada alone produced $28,130,350, while in 1883, owing to the decline of the Comstock mine, it produced only $5,430,000; Utah, in 1883, produced $5,620,000, and Colorada $17,370,000, and is therefore as she has been since 1879, the great silver-producing State.

The following Tables, prepared from the reports of the director of the United States Mint, give the production of each of the silver-producing States from 1876 to 1884.

VALUE OF SILVER PRODUCED IN THE UNITED STATES.

STATES.	1876.	1877.	1878.	1879.	1880.	1881.	1882.	1883.	1884.
	$	$	$	$	$	$	$	$	$
Alaska	...	...	...	...	...	...	...	...	...
Arizona	500,000	3,000,000	3,550,000	2,000,000	7,800,000	7,300,000	7,500,000	5,200,000	4,500,000
California	1,000,000	2,373,389	2,400,000	1,100,000	870,000	750,000	845,000	6,460,000	3,000,000
Colorado	4,500,000	5,394,940	11,700,000	17,000,000	15,000,000	17,160,000	16,500,000	17,370,000	16,000,000
Dakota	...	...	10,000	70,000	60,000	70,000	175,000	150,000	150,000
Georgia	...	...	...	...	...	...	...	1,000	...
Idaho	250,000	200,000	650,000	450,000	1,100,000	1,300,000	2,000,000	2,100,000	2,720,000
Maine	...	...	...	...	...	5,000	...	...	...
Michigan	200,000	100,000	780,000	...	...	...	...	...	...
Montana	750,000	1,669,635	2,225,000	2,500,000	2,300,000	2,630,000	4,370,000	6,000,000	7,000,000
Nevada	26,000,000	28,130,350	12,560,000	10,900,000	8,860,000	7,060,000	6,700,000	5,430,000	5,600,000
New Mexico	500,000	500,000	600,000	425,000	270,000	275,000	1,800,000	2,845,000	3,000,000
North Carolina	...	...	...	...	...	...	25,000	3,000	3,500
Oregon	100,000	100,000	20,000	15,000	80,000	50,000	35,000	20,000	20,000
South Carolina	...	...	...	...	...	...	...	500	500
Tennessee & Alabama	...	...	...	...	...	...	...	...	...
Utah	5,075,000	5,208,000	6,250,000	4,740,000	5,710,000	6,400,000	6,800,000	5,620,000	6,800,000
Virginia	...	...	...	...	...	...	...	...	5,000
Washington T.	50,000	25,000	20,000	...	...	...	...	500	1,000
Wyoming	...	...	...	...	...	...	...	...	...
Other States	25,000	25,000	47,000	...	50,000	...	...	...	...

The following Table, compiled from the same source, gives the total gold and silver of the different States from 1877 to 1884.

TOTAL VALUE OF GOLD AND SILVER PRODUCED IN THE UNITED STATES.

STATES.	1876.	1877.	1878.	1879.	1880.	1881.	1882.	1883.	1884.
	$	$	$	$	$	$	$	$	$
Alaska	...	...	...	6,000	7,000	15,000	150,000	300,000	200,000
Arizona	800,000	3,500,000	4,350,000	2,400,000	8,570,000	8,360,000	8,565,000	6,150,000	5,430,000
California	16,000,000	17,634,068	20,000,000	18,600,000	19,870,000	18,950,000	17,645,000	15,580,000	16,600,000
Colorado	7,500,000	8,761,344	14,925,000	20,200,000	18,400,000	20,460,000	19,860,000	21,470,000	20,250,000
Dakota	2,000,000	3,000.000	2,430,000	3,670,000	4,560,000	4,070,000	3,475,000	3,350,000	3,450,000
Georgia	100,000	100,000	90,000	120,000	150,000	125,000	250,000	200,000	137,000
Idaho	1,750,000	1,350,000	1,850,000	2,430,000	3,030,000	3,000,000	3,500,000	3,500,000	3,970,000
Maine	...	...	...	...	...	5,000	...	...	...
Michigan	200,000	100,000	780,000	...	...	...	...	...	...
Montana	3,950,000	3,930,146	4,725,000	4,900,000	4,800,000	4,960,000	6,920,000	7,800,000	9,170,000
Nevada	44,000,000	47,676,863	21,560,000	15,700,000	11,560,000	9,310,000	8,750,000	7,950,000	9,100,000
New Mexico	675,000	675,000	725,000	555,000	390,000	460,000	1,950,000	3,125,000	3,300,000
North Carolina	100,000	150,000	90,000	95,000	75,000	115,000	215,000	170,000	160,500
Oregon	1,100,000	1,100,000	1,170,000	1,105,000	1,080,000	1,150,000	865,000	680,000	680,000
South Carolina	...	...	...	15,000	18,000	35,000	25,000	57,000	57,500
Tennessee & Alabama	...	...	...	...	2,000	5,000	...	...	81,000
Utah	5,425,000	5,600,000	6,825,000	4,950,000	5,910,000	6,545,000	6,990,000	5,760,000	6,920,000
Virginia	50,000	...	...	10,000	11,000	10,000	15,000	6,000	2,000
Washington T.	350,000	325,000	95,000	410,000	100,000	120,000	120,000	80,500	89,000
Wyoming	...	...	...	20,000	7,000	5,000	5,000	4,000	6,000
Other States	50,000	50,000	97,000	14,000	60,000	...	...	17,500	...

The following Table, compiled from the same sources, gives the total amount of silver produced yearly in the United States from 1858 to 1884, and also the total value of the gold and silver for the same years.

Year.	Silver.	Total Gold and Silver.	Year.	Silver.	Total Gold and Silver.
	$	$		$	$
1858 . .	500,000	50,500,000	1871 .	23,000,000	66,500,000
1859 . .	100,000	50,100,000	1872 .	28,750,000	64,750,000
1860 . .	150,000	46,150,000	1873 .	35,750,000	71,750,000
1861 . .	2,000,000	45,000,000	1874 .	37,324,594	70,815,496
1862 . .	4,500,000	43,700,000	1875 .	31,727,560	65,195,416
1863 . .	8,500,000	48,500,000	1876 .	38,783,016	78,712,182
1864 . .	11,000,000	57,100,000	1877 .	39,793,573	86,690,963
1865 . .	11,250,000	64,475,000	1878 .	45,281,385	96,487,745
1866 . .	10,000,000	63,500,000	1879 .	40,812,132	79,711,990
1867 . .	13,500,000	65,225,000	1880 .	39,200,000	75,200,000
1868 . .	12,000,000	60,000,000	1881 .	42,000,000	78,600,000
1869 . .	12,000,000	61,500,000	1882 .	43,000,000	77,700,000
1870 . .	16,000,000	66,000,000	1883 .	46,800,000	79,800,000
			1884 .	48,800,000	79,600,000

Such enormous productions of the precious metals have not been without their influence on the relative value of gold and silver in other countries. The United States is one of the largest producers of the precious metals, notwithstanding, as the statistics show, that there has been a gradual falling off in the production of gold, and the highest limit of silver appears to have been in the year 1883. Notwithstanding the continued decrease in the production of the Comstock from its maximum in 1877, nearly $22,000,000, the increase in Colorado and other States has more than compensated for it.

The amount of gold consumed in the United States for purposes of art and ornament during the year 1879 was larger than for several previous years. The following Table from the Report of the Director of the Mint, which is a mine of information for those interested in the production and distribution of the precious metals, gives the returns of the New York Assay Office for that year :

BARS MANUFACTURED.*

—	Gold.	Silver.	Total.
	$	$	$
From United States coin (defaced)	4,929	982	5,911
Foreign coin	260,222	72,668	332,890
,, bullion	1,007,400	278,622	1,286,022
Domestic ,, ...	2,988,422	3.863,126	6,851,548
Plate, &c.	394,871	144,992	539,863
Total	4,655,844	4,360,390	9,016,234

From the whole United States this amount is much larger; but leaving out the foreign bullion altogether, the following Table gives the estimate of the total gold and silver used in the whole United States for industrial purposes during the year 1881:

—	Silver.	Gold.
	$	$
Domestic bullion	4,000,000	5,000,000
United States coin	600,000	2,500,000
Plate, foreign bullion, and coin . .	400,000	2,500,000
Amount consumed . . .	5,000,000	10,000,000

The consumption of the precious metals for purposes of art and ornament has been the subject of estimates by many distinguished statisticians, but at the best can only be approximated. In 1827 Humboldt placed it at 375,000 ounces, or one-fifth of the world's production at that time. In 1822 Lowe estimated it at two-thirds. William Jacob estimated it at 988,000 ounces, which was double the average annual production between 1821 and 1830. Dr. Soetbeer, of Germany, gives the following Tables of the consumption of the precious metals for jewelry and other industrial purposes in the various countries of the world:†

* Report of the Director of the Mint, 1880, p. 19.

† "Engineering and Mining Journal," vol. xxxii., p. 183.

GOLD.

COUNTRIES.	Consumption, in Ounces.	Reduction by Old Material used.	Total Consumption.
United States	529,000	10 per cent.	476,000
Great Britain	703,000	15 per cent.	598,000
France	739,000	20 per cent.	591,000
Germany	518,000	20 per cent.	412,000
Switzerland	529,000	25 per cent.	397,000
Austria	102,000	15 per cent.	87,000
Italy	212,000	25 per cent.	159,000
Russia	106,000	20 per cent.	85,000
Other countries	176,000	20 per cent.	141,000
Total	3,614,000	...	2,946,000

SILVER.

COUNTRIES.	Consumption, in Ounces.	Reduction by Old Material used.	Total Consumption.
	$	$	$
United States	4,233,000	15 per cent.	3,789,000
Great Britain	3,175,000	20 per cent.	2,540,000
France	3,528,000	25 per cent.	2,646,000
Austria-Hungary	1,411,000	20 per cent.	1,129,000
Switzerland	1,129,000	25 per cent.	847,000
Italy	882,000	25 per cent.	662,000
Russia	1,411,000	20 per cent.	1,129,000
Germany	3,528,000	25 per cent.	2,646,000
Prussia	1,870,000	...	1,411,000
Total	21,167,000	...	16,799,000

The following Tables, taken from the Report of the Director of the United States Mint for 1884 (pp. 62 and 63), give the character and used of the bullion deposited in and withdrawn from the New York Assay Office in 1884.

PRECIOUS METALS used in the Arts and Manufactures in the United States.

DEPOSITS	Gold.	Silver.
	$	$
Of United States coin	1,171 25	1,833 73
Of foreign coin	97,465 06	63,184 30
Of foreign bullion	250,225 14	359,823 81
Of plate, &c.	713,099 05	166,625 67
Of domestic bullion	3,553,157 64	4,614,529 46
Total	4,615,118 14	5,205,996 97
Large gold bars exchanged for gold coin and redeposited for small bars, $1,260,942.27, less the charges and fractions paid in gold coin	1,259,893 58	
Total	5,875,011 72	

TABLE SHOWING THE VALUE AND CHARACTER OF THE GOLD AND SILVER USED IN THE ARTS AND MANUFACTURES DURING THE CALENDAR YEAR 1883.

GOLD.

Manufactures.	Number Manufacturing.	United States Coin.	Stamped United States or Refinery Bars.	Old Jewelry, Plate, and other Old Material.	Foreign Coin.	Native Grains, Nuggets, &c.	Wire or Rolled Plate.	Total Gold.
		$	$	$	$	$	$	$
Watch-cases	32	575,812	2,976,550	38,101	1,508	520	5,817	3,598,308
Watch-chains	14	374,997	286,884	1,907	600	135,410	27,202	827,000
Dental supplies . . .	7	700	33,437	3,775	...	...	...	37,912
Pens	14	14,578	90,325	6,100	5,227	2,134	27,560	145,924
Instruments	45	68	...	3,568	...	621	942	5,199
Leaf	51	178,424	792,551	57,498	6,816	6,700	42,835	1,084,824
Plate	219	379,291	67,928	5,500	590	8,933	66,626	528,868
Spectacles	41	192,400	7,169	8,830	1,315	4,987	727	215,428
Chemicals	27	7,438	7,685	3,551	550	207	12,180	31,611
Jewelry and watchmakers' supplies	11	24,498	13,983	9,123	...	1,569	30,054	79,227
Jewelry and watches . .	2,273	3,125,738	2,861,149	738,688	177,794	541,306	458,745	7,905,163
Total	2,734	4,875,587	7,137,661	876,641	194,400	702,387	672,688	14,459,464

TABLE SHOWING THE VALUE AND CHARACTER OF THE GOLD AND SILVER USED IN THE ARTS AND MANUFACTURES DURING THE CALENDAR YEAR 1883.

SILVER.

Manufactures.	United States Coin.	Stamped United States or Refinery Bars.	Old Jewelry Plate, and other Old Material.	Foreign Coin.	Native Grains, Nuggets, &c.	Wire or Rolled Plates.	Total Silver.	Total Silver and Gold.
	$	$	$	$	$	$	$	$
Watch-cases	35,200	1,777,193	31,937	219	1,000	50	1,845,599	5,443,907
Watch-chains	524	14,768	...	...	6,790	1,462	23,544	850,544
Dental supplies	450	6,060	...	...	...	228	6,738	44,650
Pens	216	4,254	100	1,655	505	...	6,730	152,654
Instruments	931	3,752	693	755	864	6,995	13,990	19,189
Leaf	11	22,697	4,107	300	835	18,933	46,883	1,131,707
Plate	16,856	1,710,515	40,761	7,690	8,495	281,977	2,066,294	2,595,162
Spectacles	3,631	16,461	1,254	205	250	1,981	23,782	239,210
Chemicals	9	375,429	35,554	500	1,580	3,347	416,419	448,030
Jewelry and watchmakers' supplies	245	4,806	800	...	1,505	975	8,331	87,558
Jewelry and watches	158,564	616,237	106,745	142,949	49,733	23,992	1,098,220	9,003,383
Total	216,637	4,552,172	221,951	154,273	71,557	339,940	5,556,530	20,015,994

Character and Value of the Precious Metals reported by Manufacturers, Jewellers, and Others, used by them during the Calendar Year 1883.

Character.	Gold.	Silver.	Total.
	$	$	$
United States coin . . .	4,875,587	216,637	5,092,224
Stamped United States or refinery bars	7,137,761	4,552,172	11,689,933
Old jewelry, plate, and other materials	876,641	221,951	1,098,592
Foreign coin	194,400	154,273	348,673
Native grains, nuggets, &c. . .	702,387	71,557	773,944
Wire, or rolled plate . . .	672,688	339,940	1,012,628
Value . . .	14,459,464	5,556,530	20,015,994

Other estimates give the entire consumption of the precious metals in Europe and America for industrial purposes in 1880 as from $45,000,000 to $55,000,000 in gold, and from $25,000,000 to $30,000,000 in silver.

From 1831 to 1880 the estimated consumption of gold for industrial purposes was 73,000,000 ounces, or 32.6 per cent. of that produced. For silver it was 511,000,000, or 25.2 per cent.

Of the world's product of bullion it is estimated that one-third is used up and lost in the wear and tear of coins and articles made for use or ornament, one-third is used for manufacturing purposes, and one-third goes to supply the increased demands of trade. The amount lost by the abrasion of coins is shown by the fact that the average life of an English sovereign is eighteen years, by which time the coin has lost three-quarters of a grain and is no longer a legal tender. Dr. Soetbeer states that the annual loss from this source in civilised countries reaches 28,000 ounces of gold and 1,600,000 ounces of silver.

The following Table* shows the amount of gold and silver produced in the world in the years 1877 to 1884:

* Report of the Director of the Mint for 1884.

AMOUNT OF GOLD AND SILVER PRODUCED IN THE WORLD IN THE YEARS 1877 TO 1884.

Countries.	1877.		1878.		1879.		1880.	
	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
	$	$	$	$	$	$	$	$
United States	47,897,390	39,793,573	51,206,360	45,281,385	38,899,858	40,812,132	36,000,000	39,200,000
Russia	27,226,668	467,844	27,967,697	448,016	26,584,000	415,676	28,551,028	473,519
Australia	29,018,223	...	29,018,223	...	29,018,223	...	28,765,000	227,125
Mexico	996,898	27,018,980	996,898	27,018,940	989,161	25,167,763	989,160	25,167,763
Germany	204,697	6,135,877	205,361	6,938,073	205,361	6,938,073	232,610	7,730,617
Austria	1,196,278	2,119,948	1,196,278	2,161,515	1,062,031	2,002,727	1,094,596	1,994,880
Sweden	2,658	54,038	6,001	52,708	1,994	62,435	3,323	54,527
Norway	...	188,052	...	166,270	...	166,270	...	184,360
Italy	72,375	17,949	72,375	17,949	72,375	17,949	72,375	17,949
Rest of Europe	...	2,078,380	...	2,078,380	...	2,078,380	4,918	3,167,661
Argentine Republic	78,546	420,225	78,546	420,225	78,546	420,225	78,546	420,225
Colombia	4,000,000	1,000,000	4,000,000	1,000,000	4,000,000	1,000,000	4,000,000	1,000,000
Rest of South America	1,993,800	1,039,190	1,993,800	1,039,190	1,993,800	1,039,190	1,095,101	16,081,747
Japan	265,840	706,649	295,746	728,846	466,548	916,400	466,548	916,400
Africa	1,993,800	...	1,993,800	...	1,993,800	...	1,993,80	...
Venezuela	...	...	...	...	...	...	2,274,692	...
Dominion of Canada	...	...	...	...	...	...	815,089	68,205
France	...	...	...	...	...	...	...	...
Peru	...	...	...	...	...	...	...	...
Total	113,947,173	81,040,665	119,031,085	87,351,497	105,365,697	81,031,220	106,436,786	96,704,978

AMOUNT OF GOLD AND SILVER PRODUCED IN THE WORLD IN THE YEARS 1877 TO 1884—*continued.*

Countries.	1881.		1882.		1883.		1884.	
	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
	$	$	$	$	$	$	$	$
United States	34,700,000	43,000,000	32,500,000	46,800,000	30,000,000	46,200,000	30,800,000	48,800,000
Russia	24,371,343	332,198	23,867,935	323,427	23,867,935	323,427	21,818,304	388,000
Australia	30,690,000	164,983	28,943,217	102,878	26,500,000	89,418	28,551,101	115,960
Mexico	858,909	27,675,540	936,223	29,237,798	955,639	29,568,576	1,183,137	27,257,885
Germany	232,610	7,771,304	249,890	8,934,652	303,722	9,589,300	368,853	10,311,659
Austria-Hungary	1,240,808	1,303,280	1,050,068	1,958,224	1,088,615	2,024,645	1,101,707	2,054,070
Sweden	665	48,875	11,298	62,350	24,590	65,800	12,627	75,472
Norway	...	199,987	...	244,954	...	234,645	...	265,490
Italy	72,375	17,949	72,375	17,949	72,375	17,949	72,375	17,949
Spain	...	3,096,220	...	3,096,220	...	3,096,220	...	148,000
Turkey	4,918	71,441	6,646	89,916	6,646	89,916	6,646	89,916
Argentine Republic	78,546	420,225	78,546	420,225	78,546	420,225	78,546	420,225
Colombia	4,000,000	1,000,000	3,856,000	760,000	3,856,000	760,000	3,856,000	760,000
Bolivia	72,375	11,000,000	72,375	11,000,000	72,375	16,000,000	72,375	16,000,000
Chili	128,869	5,081,747	163,000	5,325,000	163,000	5,325,000	163,000	5,325,000
Brazil	741,694	...	741,694	...	632,520	...	632,520	...
Japan	466,548	916,400	632,520	877,772	170,270	877,772	170,270	877,772
Africa	1,993,800	...	1,993,800	...	1,993,800	...	1,993,800	...
Venezuela	2,274,692	...	2,595,077	...	3,338,058	...	3,338,058	...
Dominion of Canada	1,094,926	68,205	1,094,926	68,205	954,000	68,205	954,000	68,205
France	...	...	...	594,053	...	164,275	...	264,275
Peru	...	...	119,250	1,908,000	119,250	1,908,000	119,250	1,908,000
Total	103,023,078	102,168,354	98,984,840	111,821,623	94,197,341	116,923,873	95,292,569	115,147,878

These Tables show that the United States is by far the greatest producer of the precious metals, Russia being the only one which produces anything like as much gold, and Mexico the only one that approaches it in silver.

The amount of precious* metals sent to the East, the greater part of which goes to India, has been estimated by Dr. Soetbeer as:

	Gold, in Ounces.	Silver, in Ounces.
1831-1840	35,000	7,750,000
1871-1880	423,000	38,000,000
1831-1880	19,700,000	1,376,000,000

In the period from 1871-1880, which is most reliable, the consumption of gold by this means was 47,000 ounces, and of silver 4,200,000 ounces. In India alone the imports in the last forty years have exceeded the exports of these metals by $400,000,000, of which only $8,000,000 have been coined as money.

The amount of the precious metal hoarded or put out of circulation either as objects of art or ornament is becoming greater with every decade. It appears from the data given, page 52, that the total annual consumption of the precious metals for purposes other than coinage is about 3,600,000 ounces of gold and 21,000,000 of silver. It has been estimated that the entire amount of gold now in the world is only equal to that which has been produced in the last twenty-five years, and that of silver to that produced in the last eighty years. No one has as yet been able to satisfactorily explain what has become of all the rest of the precious metals.

Only an estimate can be made of their wear and tear, which is an irretrievable loss, either in the abrasion of coin or in the use of leaf or of the pure metals for plating purposes. Add to this the amount lost in lead, copper, and other metals, which do not contain enough of it to separate, and it is not a matter of surprise that, notwithstanding the enormous yearly increase, the estimate of the total amount supposed to exist in the world during any period is not perceptibly greater.

In all the methods for the extraction of the precious metals, there are considerable losses. With the perfection of processes

* "Engineering and Mining Journal," vol. xxxii. p. 183.

the main object is to reduce them, or else to cheapen the labour of extracting the ores. These losses are greater than is usually supposed, because as a general rule systematic assays of the tails are not made. Yet it is known that the tails contain precious metals, and they are sometimes re-worked with profit, especially those from the silver mines. An interesting investigation was made some years ago, the results of which are given below,* showing the great loss in some of the mills.

——	Yield of Ore in the Mill.	Remaining in Tails.		Total in Tails.
		Gold.	Silver.	
	$	$	$	$
At the mill	18 60	10 04	3 14	13 18
Same tailings 350 ft. from mill	18 60	5 02	3 92	8 98
Average yield of 150 tons .	3 50	13 55	6 28	18 83
,, ,, ,, .	3 50	8 79	62 8	15 07
Slime from end of sluice 310 ft. long . . .	...	56 00	33 30	89 00

It was also found that water from the mills three-fourths of a mile below them contained in suspension, as an average of twelve assays, $0.018 per gallon. There were in this locality 576,000 gallons of this water flowing away in 24 hours, or a loss of $339.84. It was estimated that the annual loss of two mills working 250 days in the year was $84,960. From these and similar data the conclusion is drawn, that the loss is between 50 per cent. and 60 per cent. of the total yield of the ore.

It is a matter of great interest to ascertain what the cause of these losses is, in order to learn how far they are capable of remedy. The first of these is undoubtedly a desire to get the largest possible output from the mill. This makes the ore too coarse to have all the gold and silver amalgamate, as part of it may not be released from the gangue. It would be much better to get the output by a more careful sizing of the ore, not forcing the stamp to do the work of a Blake's crusher, and not sending to the mortars any ore fine enough to pass the screens. This

* Report of the United States Mining Commissioner, 1872, p. 17.

is a matter of some importance, for it has been found, with all kinds of stamps using screens, that it takes just as long to get crushed ore which has already passed the screens out of the mortar, as it does to crush and force it out. Too fine crushing is also quite as bad, for it produces "float," and is quite likely to put the precious metals in a condition in which they will not amalgamate.

Supposing that the losses which result from improper working do not exist, there are a few causes of loss which do not always amount to much, but which, in the early days, were a source of considerable loss. It has been found that any holes in the castings of the stamps, pans, &c., will attract the amalgam, and that it will even be carried into holes deep in the interior of the piece. This was a source of profit in the early days to those who recovered the precious metals when the worn-out castings were melted. Another loss may be in cleaning the plates by taking off the amalgam too thoroughly. It is a well-known fact that new plates do not act as readily as old ones; the difference is so great, that when the mills can afford it, new plates are coated with gold or silver amalgam. Gold and silver will go much quicker to amalgam than to mercury. Too slow a current of water will keep the surface of the plates covered with a film of sand; a too rapid current will prevent the gold from being caught. If the gold is attached to a piece of the gangue rock which is relatively large, the specific gravity may be so reduced as to prevent the particles from coming in contact with the mercury. If the blankets are left too long without washing, so that the hairs become charged, the fine particles of gold are lost. If all these causes of loss are avoided, there are still others. For if the mercury is not kept clean or made so by chemicals the "quick," having an extremely thin film upon it, does not act upon the gold or silver. Exactly the same effect is produced to a small extent when the rock is soapy, as is the case with some of the hydrated magnesian and aluminous rocks. If there are too few amalgamating machines, if the sluices are too short, there is also a loss. A very important source of loss is the flouring of the mercury from too rapid motion, or from the too free use of chemicals. In such cases steam may be used, if it is live

steam fresh from the boiler, which prevents flouring by the expansion of the globules. If steam from the engine is used as a matter of economy, it often increases the loss, as very minute particles of grease are always carried off with it, which coat the mercury, and prevent its coming in contact with the gold. The cause of the losses on the concentrates has already been discussed.

CHAPTER I.

SILVER ORES AND SAMPLING.

The gold and silver in the ores of the United States are confined to the region lying between the Rocky Mountains and the Pacific Coast Range. The eastern part of the United States furnishes very little material which can be treated for gold and silver, except the alluvial sands of the Southern States, and a few copper ores which cannot be amalgamated, and which have never been smelted.

The eastern coast of Massachusetts, and of Maine, has furnished small quantities of tetrahedrite containing considerable silver, but the amount has been very uncertain, and the grade is low; the treatment is difficult, and they have been abandoned or worked only in a very small way.

The great silver-producing region of the United States lies, therefore, in the Rocky Mountains and west of them. The ores themselves comprise every possible variety both in their mineralogical constitutents and in their richness. They will generally be divided according to their mode of occurrence, those in which the silver and gold are combined with some other metal which requires smelting for their separation, those in which the ore may be treated directly or indirectly with mercury, and those which must be leached. The ores which are to be smelted are composed of ores containing lead, which are themselves divided into the oxidised ores like those of Leadville and of Utah, which are smelted directly without any preparation, those which are composed of sulphurets which are either free from or mixed with other metals, such as zinc and copper, and must be roasted, and pure or impure copper ores containing the precious metals in variable quantities to which any other ores containing gold and silver may be added. The furnaces in which lead ores are used are exclusively of the

shaft furnace type. The reverberatory furnace is not generally applicable to the conditions under which these ores are found. Sulphurets, if they are pure, are simply roasted and treated in the shaft furnace, or, if they are impure, an effort is made to separate the impurities as far as possible by hand-picking and further concentration, and then the materials are smelted in the shaft furnace, and if much copper is present the result will be a copper matte which will always contain more or less silver and gold, and the lead which contains the rest of the precious metal is treated by itself. Usually the copper mattes produced in the fusion of lead ores are sent abroad for treatment. They are, when made from shaft furnaces, never treated in the works where they are produced. They contain a considerable percentage of the gold and silver. The Boston and Colorado Works, at Denver, and the works of Pope, Cole and Co., in Baltimore, treat almost exclusively copper ores containing gold and silver, in reverberatory furnaces, and separate the gold and silver from a copper matte.

The lead in which the silver and gold is contained is universally treated by the zinc desilverisation process. No other process of separation has been used to any extent* in the United States. An establishment for patisonnage was erected in Bloomfield, New Jersey, previous to the war, but it never was successful, and it would be difficult to find even the ruins at this day. The process by desilverisation has been elaborated so that it is much more economical to concentrate the silver in the lead by this means than to attempt to concentrate it in any other way. It has been successfully used in a large number of establishments. The steps by which the present process has been arrived at it will be given in detail.

When the lead ores contain copper it does not of necessity make any complication in the process further than the production of a copper matte, thus dividing the silver contents of the ores. This copper matte can be easily separated from the lead and concentrated by subsequent roasting and fusion, while the small amount of copper contained in the lead can be easily refined out.

When the gold and silver ores are found in copper ores, pro-

* The Rozan process is used at Eureka, Nevada.

perly speaking, those which contain lead enough to be smelted for lead ores, are separated by hand-picking where that is practicable. Where it is not practicable the ores are simply added to those containing copper, care being taken to add only such quantities as will not interfere with the treatment in the furnaces, and they are then treated as ordinary copper ores, the lead being entirely lost. This is done in the Boston and Colorado works, which will be described in detail. The silver ores, properly speaking, are treated according as they contain more sulphur, that is, are "heavy," or those which are free milling. The former must be crushed and roasted. This is done in a large number of furnaces which will be described, the roasting being the chlorurising roasting in order to get the silver into such a condition that it will be easily attacked by the mercury. Free-milling ores are treated without roasting; they are simply crushed. This crushing has given rise to a very elaborate machinery which is used for the purpose, which will be described in detail. The machinery grew little by little from the old German methods of crushing, until there ceased to be any resemblance between the type of the stamp from which they originated, and the modern mill. Until within a very few years it was supposed, on account of some experiments which have been made in Lake Superior on ores not adapted to them and which had failed, that rolls could not be used in the United States. We are gradually coming back from that opinion, and the stamp mill, except under certain very peculiar conditions, has probably seen its best days.

When, however, the ores are not adapted for any of these processes, if they are rich enough they are treated by lixiviation by the Von Patera's or some kindred process, they should not however contain any considerable quantity of base metal. Until the beginning of the year 1884, leaching had been applied to ores containing only a very small quantity of lead, because the lead was soluble in the dissolving medium. Since that time Mr. E. H. Russell has invented a process which seems likely to make an entire revolution in this respect. As by this process all the lead can be precipitated and the liquor regenerated while the base metals, such as copper, can all be saved, there seems to be a great future for this process. This renders available a very large quantity of ores

so poor that they cannot be treated by any of the other processes, and will probably make it possible to treat very large quantities of tails where these have been stocked in tail reservoirs, as the cost of extraction is so very small that ores of very low grade can be treated.

It will thus be seen that Western gold and silver ores are, for metallurgical purposes, divided into four classes:

1. Those which contain copper enough to be smelted for copper from which the gold and silver is extracted in the wet way, as is the practice of the Boston and Colorado works;
2. Those in which there is a large quantity of lead, which can be smelted for lead, and the gold and silver extracted from it;
3. Ores in which there is neither copper or lead enough to allow of a process of smelting, but which can be treated in pans, these ores being "free milling" if they require no metallurgical treatment, or "rebellious" if they have to be roasted with or without the addition of salt;
4. Ores which do not contain enough either of lead or copper for smelting, which are poor both in silver and gold, contain large amounts of sulphur, arsenic, and antimony, and cannot be treated in many places in the West by any of these processes, and which can only be treated by leaching.

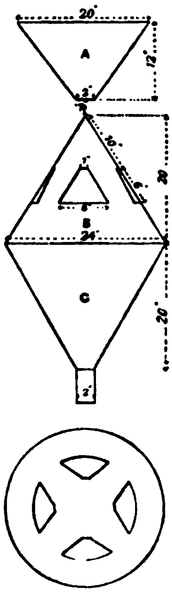
Roasting with salt would convert the base metals as well as the silver into chlorides, and would give in the amalgamation a very base bullion. The expense of the process would be so great that the margin of profit would be very small, the reason being that for the ordinary process of pan amalgamation, which is the only one suitable for ores containing small amounts of the base metals, and poor in silver and gold, the cost of a plant for milling is so large as not to justify the expense of treating such ores. The electrolytic processes which have been partially successful in Europe have not, strictly speaking, passed the experimental stage here. While in the near future they will undoubtedly be used, it is hardly possible to consider them now. Works can only be erected near great centres of population, or where the amount of ore to be treated is so very large

that it would justify a very expensive plant. When the trial period is passed, this method will undoubtedly be applicable to many ores whose treatment is not now even discussed as possible. There are very large quantities of low-grade ores containing about 10 to 20 ounces of silver, with little or no gold, to the ton, which might be treated if a not very expensive plant could be used.

All the ores, whether they are to be treated by the mine or by custom mills, or whether they are to be sold, require to be assayed; in order to do this they must be sampled, and as mine sampling on coarse ore cannot be relied upon for determining the value of the precious metals contained, the ore must be crushed in order to get it into its proper condition for making the assay. The taking of the sample in such cases is difficult, because in many of the ores there are either metallic materials like silver and gold, or there are mineral compounds like the sulphide, the chlorides, and bromides of silver, which are themselves malleable, and for this reason render the assay difficult. In such a case as this, the sifting must be done very carefully, and whatever remains behind on the sieve as malleable material must be assayed as a whole and not sampled, while that which passes through may be sampled and the total value of the ore ascertained by calculation. As this sampling is extremely difficult when done by hand, there are usually some appliances by which it is done more or less mechanically, so that for the sample to be taken for the laboratory about one-tenth of the total ore passing through the stamps or rolls is collected from which to take the assay sample. As far as possible this is done automatically, and there are very many ingenious contrivances in the different mills for doing this work upon the ore. In some dry crushing mills the sample is taken from under the crusher by causing a box to pass under the mouth at certain intervals, remaining there only just long enough to be filled, and then being withdrawn and dumped. In others it is done in front of the stamp screens by means of the endless chain which conveys the ore to the hopper from which it is charged into the roasting furnace. The conveyor is so arranged that every tenth bucket tips into a box instead of being carried up by the elevator. In other cases a wheel with buckets is used,

the bottoms of a given number of which are removed, so that the ore during the time of its passage falls into a collecting box below, or a single box rotating on a shaft at a given velocity is placed directly in front of the stamp and discharges its contents as it revolves into a receptacle from which it is from time to time collected. A given proportion of the whole quantity of ore is thus collected in a stated time. It follows necessarily in such a case as this that the ore must be dry or dried before this can be done. Below is given a description of the very ingenious way by which this sampling was formerly done in two of the Colorado mills at the Lebanon Mine and Bennett's Mill.

At the Lebanon Mine Company's Works, the ore from the crusher is raised by an endless chain, and is charged into a bin from the bottom of which a funnel A (see annexed diagram), 12 in. high and 20 in. in diameter at the top, with an opening 2 in. in



Plan of B.C.

diameter at the bottom, discharges the ore over the sampler B C. This funnel is covered with a coarse screen to keep out pieces of ore of too large a size, pieces of wood, leather, or any other material which should not pass over the sampler but which

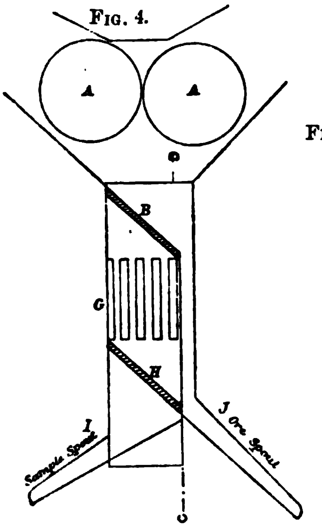
might be carried up by the chain. The sampler consists of a cone B, the apex of which is exactly in the centre of the discharge tube of the funnel A, and 2½ in. below it. The cone B is 20 in. high, and 24 in. in diameter at its base. At this point another cone C is securely fastened to it by its base, and from the small end of the lower cone a discharge pipe 2 in. in diameter leads to a wooden receptacle below. The upper cone has four holes at equal distances, these commencing at 10 in. from its apex. They are 1 in. wide at the top, and 8 in. at the bottom, and 6 in. long. All of the ore discharged into the upper funnel falls over this upper cone, and part of it passes through these holes, and falls into the receptacle, where it is collected. The size of these holes is such that the sample obtained will be 8 per cent. of the ore. They can, however, be arranged to give a larger or smaller supply by increasing or diminishing their size. It is necessary to have a considerable number of extra caps for the cone, as the ore falling constantly upon them wears them out rapidly, and they must be replaced. All of the ore which is discharged from the cone goes into a bin to be bagged for shipment.

When the whole of the ore to be sampled has passed over the sampler, the first sample collected is thrown back again into the upper hopper, and this is repeated several times until it is reduced to about 40 lb. It is then passed through a sieve of 40 to the inch, and the ordinary sample for assay taken of it. All the rest of the first sample is bagged to be sold or treated.

At Bennett's Mill the sample is taken somewhat differently. The ore is raised to two Cornish rolls A, Figs. 4 and 5, from which it is discharged into a hopper. This hopper has a trough at the bottom, which is provided with a slanting shelf B at an angle of 50 deg., which is divided into nine equal parts, six of which open on to another inclined shelf G. At the end of the shelf B, an opening one-half times as wide as the openings in the shelf leads down to the discharging trough J. The shelf G has the same angle of inclination as the shelf B, but is at right angles to it, and is divided into ten equal parts with five openings. The ore which passes over this shelf is discharged into the slide J. What passes through it is discharged on to another shelf H at right angles to G, but with the same inclination and

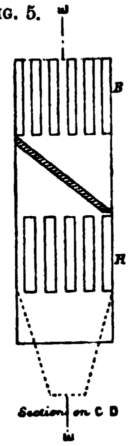
parallel to the shelf B. This shelf is also divided into ten equal parts with five openings. What passes over it goes to the dis-

Fig. 4.



Section on E F.

Fig. 5.



Section on C D

charge spout J, and what goes through it goes to the sample spout I, where it is collected, and the assay sample taken from it.*

The whole of the ore from the sampler is carefully made into a conical pile. From different parts of it with a motion towards the centre, and up and down from the bottom and diagonally across, samples are taken with a scoop, Fig. 12. When the pile is very much broken and has lost its shape, it is turned over and reformed, and the operation commenced again, and continued until the material collected amounts to a pailful. It is then poured, for the convenience of handlingit, into an ordinary miner's gold pan, Fig. 6, which is 16 in. in diameter at the top, 10 in. at the bottom, 3 in. high, and is then turned on to a box made of tin, which is very often arranged with a handle, so as to be used as a shovel to take a sample in a pile, 17 in. square, divided into 17 divisions 1 in. wide, Fig. 7. Eight of these divisions have bottoms and catch the ore, and nine are open and allow the ore to fall on to the floor. When the box is filled with ore it is

* Another ingenious sampling machine is described in the School of Mines Report, vol. iii., p. 257.

made even on the top, the residue falling from the sides, if any, being carefully collected and put back into the pan, the box is then lifted, and what remains on the floor is returned to the ore bin. What remains in the box is put on one side. This operation is repeated until the sample is exhausted. What has been collected in the boxes is now put through another box of the same kind, Fig. 8, which is 12 in. square, 3 in. high, and has 23 divisions, ten of which catch the ore. What remains in the box is put on one side as before, and the rest returned to the ore bin. The sample collected is thrown on to a sieve, which has a wooden frame 16 in. by 12 in., and 4½ in. high. This sieve is three to the inch. What will not pass the sieve is broken on the cast-iron plate, which is 59 in. square and 1 in. thick, with a grinder to which a backward and forward motion is given. This grinder is cast flat, but is rapidly worn, so has to have a rounded surface, as shown in Fig. 9. The ground ore is then returned to the box shown in Fig. 8, and put through two or three times, depending on the size of the sample. It will then have been reduced to 3 lb. or 4 lb. What has remained in the box is poured on to a sieve, which has a tin frame, and is 12 in. in diameter, 3½ in. high, and has 14 meshes to the inch. What does not go through is again

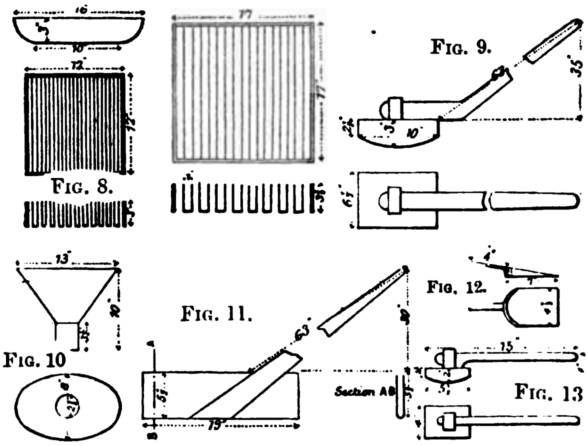


FIG. 6. FIG. 7. FIG. 8. FIG. 9. FIG. 10. FIG. 11. FIG. 12. FIG. 13.

ground, and is added to what has already passed. It is then thoroughly mixed in the pan, Fig. 6, with a suitable shovel, and

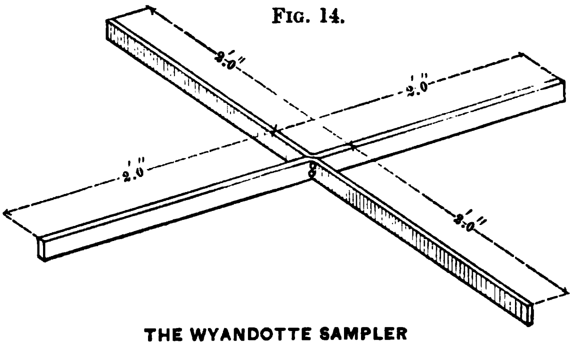
again put through the box shown, Fig. 8. The sample will now be reduced to a little less than a pint. It is emptied through the funnel, Fig. 10, into a can, and goes to the assay office. At the assay office it is still further reduced in bulk. The assay office sampler is like those shown by Figs. 4 and 5, but it is much smaller. It is 7 in. by 6 in. and ½ in. high, and has six divisions to catch the ore, and seven open ones. The sample is put through these until it is reduced one-half. It is then ground with a small grinder, Fig. 13, on a cast-iron plate 20 in. by 18 in. and 1 in. thick, which is surrounded by a wooden frame which projects about 1 in. above the top, and fits it so loosely that it can be easily removed. The ground ore is now put through a seventy-to-the-inch sieve, and is then ready for the ordinary assay. This method of sampling appears complicated from the description. It is, however, very simple, and the sample is very quickly taken. The method is very easily learned by any workman who has intelligence enough to work in any part of a sampling mill. The apparatus required is very inexpensive, and the sample is much more likely to represent the real value of the ore than samples which are taken in the usual way.

It is objected to all of these methods* that they take only a part of the stream of ore, and that the part so taken does not represent the whole. It is well known that the largest part of any crushed ore will be very much finer than the sieve through which it passes. There is therefore in a falling stream a tendency for the large and rapidly falling pieces to take the centre and the finer ores to be detained longer at the sides. The whole stream does not therefore exist as a homogeneous mass which is the principle upon which all these systems of taking samples depend. It is therefore proposed to have the stream of ore fall vertically, and so to arrange a part of the spout through which it falls that, by turning a valve automatically at given intervals, the whole of the stream of ore may be turned into a sampling box during fixed periods of time, and turned out again, so that the amount collected shall accurately represent the condition of the whole of the ore at certain intervals. This method has given excellent results.

* Trans. Am. Inst. Min. Engs., vol. xiii., p. 639.

At the Wyandotte Works, the sample is taken very simply. The ore, crushed fine with the crushers, Figs. 9 and 13, is spread evenly over an iron plate, and the sampler, Fig. 14, which is simply

FIG. 14.



THE WYANDOTTE SAMPLER

two iron bars bent at right angles and rivetted together, is put down on it, separating the ore into four parts; opposite parts are taken, a new pile made and divided in the same way, and so on until the sample is complete.

Each lot of ore brought to the works is kept separate as far as possible. They are not, however, treated separately, as this would involve too much trouble and expense. The separation is only made so as to treat material of about the same value together, or to add it, in the treatment, as different parts of the process require it. The owners are either paid for it at prices for the gold, silver and lead, which are fixed by the works or regulated by the market, or the metals when separated are delivered to the owner, a certain sum being deducted for expenses, loss and profit.

When the sampling is to be done from wet crushing the pulp is generally collected by means of a probe which is arranged so as to take the sample through the whole height of the material that has settled, or the sample is taken from the materials thrown out upon the floor in front of the pans, and when sufficient has been collected and dried it is assayed. It is more difficult to get at the exact value of the materials in the pans. When, however, for any reason it is desirable to do this a sample is generally taken from the pans themselves after the whole material has been in motion for about an hour. To do this about a

pound is taken from every pan before the mercury is added, and made into a sample. After the mercury is added, in order to know the amount of material amalgamated, the sample is taken after all the foreign material has been washed out of the pan. A certain portion of the tails is then allowed to settle and the sample taken, and these three should give concurrent results. They rarely do, however, for the reason that it is extremely difficult to take this assay without using some judgment, and wherever judgment is used the sample will of necessity be too rich. No satisfactory automatic way of taking the sample from the pans has yet been devised. It is extremely difficult to be quite sure that all the mercury has settled. It is also extremely difficult to take off with the tails, material that will not be richer or poorer than it should be by any automatic method. In some works the sample is taken from the cars, a shovel being put into a box at given intervals, and the whole, after a specified time, sampled.

In taking samples from the reverberatory furnace a few ounces are taken in different parts of the furnace every hour throughout the twenty-four, and these made into a sample. In the revolving furnaces a ladle on a long iron handle is run into the furnace every hour to catch the ore as it is thrown upon the sides during the rotation of the cylinder.

In the Stetefeldt furnace the assays can be taken through the doors below by means of a ladle upon a long handle which is run into the furnace, the operation being repeated at regular intervals until there is enough for an assay; or as all the ore falls into hoppers below, the chloruration is easily determined from the bins. Separate assay must in every case be taken from the shaft and from the flue.

CHAPTER II.

ZINC DESILVERIZATION.

When lead ores contain silver, or when it occurs in other ores in districts where lead ores can be had, they are smelted alone or together and the silver afterwards separated from the lead. The silver is either extracted on the spot or more generally sent to the East to be separated there. This material is called "base bullion," a very improper name, since it is not bullion at all, but only argentiferous or work-lead; and although this term is current in the West, it should not be adopted in technical literature. The furnaces in which the ores are smelted are almost invariably shaft-furnaces, as the ores are very silicious, and the process used is that of direct or indirect precipitation. The furnaces are usually water-jacketted and generally provided with Arendt's tap. The works which treat these ores are situated for the most part in Nevada, Utah, and Colorado. A few furnaces have been erected in the East, as at St. Louis, Mansfield Valley near Pittsburgh, and in the vicinity of New York; but, generally, it will not pay to transport the ores which come from all the Western Territories to the East, when there are works competing for them at home, unless they are exceedingly rich, or there is some special business reason why they should be treated here. The ores are therefore generally treated in the West, and the pig lead containing the silver is shipped either East to the various desilverizing works, or West to San Francisco, or treated in one of the three desilverizing works of the great West. The process used for the separation of the silver is, with a single exception, that known as zinc desilverization. The Richmond Works at Eureka, Nevada, use the Rozan process of Patinsonage, but it is

the only works which uses it. There are or have been built in the United States within the last twenty years, nineteen desilverization works. Of these eight have either closed altogether, or refine only on a very small scale at irregular intervals; the works which are now refining are situated in the following cities and towns:

Newark, New Jersey.	Omaha, Nebraska.
Mansfield, Pennsylvania.	Pueblo, Colorado.
Aurora, Illinois.	Flack's Station, Utah.
Chicago, Illinois.	Eureka,* Nevada.
St. Louis, Missouri.	San Francisco, California.
Kansas City, Missouri.	

The advantages of this process, when compared to English or German cupellation or to Patinsonage, are very great, both as to the amount of capital required for the works, the rapidity of the process, and the amount both of fuel and labour required. The process itself, since its introduction into this country, has been so much simplified, that a plant for 25 tons† can be managed by three men in 12-hour shifts, and one of 40 tons by a crew of four.

The details for the description of the process of desilverization have been taken from the Germania Works, Salt Lake, the works of the St. Louis Smelting and Refining Company, at Cheltenham, near St. Louis, Mo., and those of the Pennsylvania Lead Company, at Mansfield Valley, near Pittsburgh, Penn.

The Germania Works are situated at Flack's Station, on the Utah Southern Railroad, six miles from Salt Lake City. They treat silver-lead and also ores which they purchase in the open market. They have one shaft-furnace, and their capacity is 40 tons of argentiferous lead and 3 tons of ore a day. The value of the product of the works in copper and lead counted together, silver, and gold, in 1874, was $1,350,000, in about the proportions of 5, 6, 2. The coke used comes from Connellsville, and costs $30.00 per ton.

The works of the St. Louis Smelting and Refining Company are situated at Howard Station on the Missouri, Pacific, and the St. Louis and San Francisco Railways, a short distance from St. Louis. They have two shaft-furnaces for the treatment of ore,

* Uses the Rozan process.

† "Mineral Resources of the United States, 1883–4," p. 473.

and the residues from the refining process. The coke comes from Connellsville, and weighs 40 lb. to the bushel, costing, at Cheltenham, $10 per ton. The Illinois gas coke is also used, which costs seven cents per bushel. The lead which they treat contains about one-half of one per cent. of arsenic and antimony, so that it sometimes has to be polled. They treat from 500 to 600 tons of ore per month in addition to the bullion refining. Their daily capacity is 90 tons of lead and 15,000 ounces of silver.

The works of the Pennsylvania Lead Company treat silver and lead only which is purchased in the open market. A shaft-furnace is used for the crasses and the copper mattes. They produced in the year 1884:

Bullion treated			17,342 tons
Which produced:			
Lead	16,759 tons ...	Value ...	$1,298,363.00
Copper	68,500 pounds ...	,, ...	6,850.00
Silver	2,260,114 ounces ...	,, ...	2,485,548.00
Gold	7,789 ,, ...	,, ...	159,840.00
		Total ,, ...	$3,950,601.00

As most of the works in the United States treat ore, the prices paid for ore in the best smelting works in the West are given below:

The Colorado Smelting Company, of South Pueblo, sent out the following schedule of prices in August, 1885:

LEAD ORES.

Containing	Cents per Unit.	Per Cent. Silver.	Gold per Ounce.	Charge for Treatment per Ton.
			$	$
From 10 to 20 per cent.	30	95	18	7.00
,, 21 ,, 30 ,,	35	,,	... ...	6.00
,, 31 ,, 40 ,,	40	,,	... to ...	5.00
,, 41 ,, 50 ,,	45	,,	... ...	4.00
,, 51 ,, 60 ,,	50	,,	... 19 ...	3.00
,, 61 up	50	,,	... ...	3.00

DRY ORES.

Containing	Per Cent. Silver.	Gold per Ounce.	Charge for Treatment per Ton.
Up to 100 oz.	92	... 18	13.00
101 ,, 200 ,, ...	93	... ...	12.00
201 ,, 300 ,, ...	94	... ...	11.50
301 ,, 400 ,, ...	95	... ...	11.00
401 and upwards	95	... 19	10.50

Lead quotations based on New York market, $4.01 to $4.25 per 100 lb.

These prices are subject to change. Each decline or advance in the New York market of 25 cents per 100 lb., equals an advance or reduction of 5 cents per unit.

No pay for lead under 10 per cent.

" gold " $\frac{1}{10}$ oz.

Zinc, deduct 50 cents for each per cent. over 10 in oxidised ores ; over 5 in sulphuretted ores.

Gold when under 2 oz. $18 an ounce ; over, $19.00 an ounce.

The prices paid by the Pueblo Smelting and Refining Company in August, 1885, were as follows:

Dry Ores.*	Charge for Treatment per Ton.
Silver† below 100 oz. 90 per cent. N.Y. quotations	... $12.00
" 100 to 200 " 92 " " "	... 12.00
" over 200 " 93 " " "	... 12.00
Lead Ores.‡	
Silver† below 100 oz. 90 " " "	
" 100 to 200 " 92 " " "	
" over 200 " 93 " " "	
Lead 10 to 30 per cent. 35 cents per unit ...	... 10.00
" 30 " 40 " " "	... 9.00
" 40 " 50 " " "	... 8.00
" 50 " 60 " " "	... 7.00
" over " " " "	... 6.00

The prices paid for ore by the Smelting Works at Socorro, New Mexico, in August, 1885, were as follows:

Silver and Gold Ore.†		Charge for Treatment per Ton.
Silver below 100 oz. 90 per cent. N.Y. quotations	... Silicious ore,	$12 to 18
Do. do. 100 to 150 oz. 91 do.	... Ore containing iron,	10 " 12
Do. do. 150 " 200 " 92 do. ...		
Do. do. 200 " 250 " 93 do. ...		
Do. over 250 oz. 94 do.		

* Containing less than 10 per cent. of lead are called dry ores.

† Gold ; after deducting one-tenth of an ounce, the balance is paid for at $18 per ounce.

‡ Zinc ; 50 cents is added to charge for treatment for each per cent. over 5 per cent.

Lead.*

Silver; see above schedule.	
Lead; 40 cents per unit N.Y. quotations	Silicious ore, 10 ,, 14
	Ore containing iron, 8 ,, 10

The prices paid by the Omaha and Grant Smelting and Refining Company in 1885 were approximately as follows:

Silver and Gold Ores.

	Charge for Treatment.
Silver, 90 to 95 per cent. New York quotations daily ...	$12 to $17 per ton.

Lead Ores.

5 to 20% lead,	20 c. per unit	and	93% of	gold and	silver	is paid,	$12 to $15 per ton
21 to 30 ,,	,, 30 c.	,,	94 ,,	,,	,,	,,	10 to 12 ,,
30 to 40 ,,	,, 35 c.	,,	95 ,,	,,	,,	,,	7 to 10 ,,
41 to 50 ,,	,, 40 c.	,,	,, ,,	,,	,,	,,	4 to 7 ,,
Over 50 ,,	,, 40 to 45c.	,,	,, ,,	,,	,,	,,	0 to 4 ,,

Ores having over 300 ounces of silver are paid for at a higher rate.
Gold is paid for at $18 to $20 an ounce.
Zinc—below 10 per cent. nothing is charged; over that, 50 c. each per cent.
Copper ,, 5 ,, ,, paid; ,, $1.00 to $1.50 per unit.

The following is a sample of a purchase of ore:

Lot 683. San Juan—Ore (bulk) ...		17,555 lb.	gross.
	Moisture 1%	175 ,,	,,
	Net weight	17,380 ,,	,,

Price—Gold $18.50 per oz.
Silver $1.10¼ ,, 90%
Lead 35c. a unit.

Assay—31% lead per ton, at 35c.	=	$10.85	
95 oz. silver ,, (90%)	=	94.26	
0.20 oz. gold ,, ...	=	3.70	
		$108.81	
	Less	11.00	for treatment.
		$97.81	

17,380 lb. = 8.69 tons at $97.81 per ton = $849.95.

* Lead; below 5 per cent. nothing is paid: above 5 per cent. 35 cents a unit. Zinc; below 5 per cent. nothing is deducted; above 5 per cent. 50 cents a unit is deducted.

In every case the amounts of silicon, iron, magnesia, lime, sulphur, &c., are considered, and affect the prices; so that no general statement of the prices of ores can be made.

The Western works east of the Sierra Nevada Mountains treat ores which come principally from Utah. They are earthy carbonates and sulphates, with some galena, such as are found in Little Cottonwood and Bingham Cañons. From the former place, they contain from 10 to 40 per cent. of lead, from 70 ounces to 150 ounces of silver, 1 ounce to 3 ounces of antimony, and a trace of arsenic and zinc. From Bingham Cañon they contain 30 to 50 per cent. of lead, and 10 ounces to 20 ounces of silver. The copper in these ores is sometimes as high as 6 per cent. They are transported on an average 2000 miles, some of them being brought from New Mexico. Some argentiferous blende from Colorado contains 450 ounces of silver, 10 per cent. of lead, and 20 per cent. of zinc. When these ores have been dressed, they are made into bricks for treatment in the shaft-furnace. The Utah ores are made the base of the treatment. The works also treat argentiferous lead from all parts of the country. In Nevada and west of it they treat the ores of the district, and silver-lead from all the Territories.

The ore and argentiferous lead arriving at the works are sampled and assayed. When ore is purchased at the mine, it is sampled by the agent of the company, and assayed at the works. When the assay of the agent's sample does not agree with that of the mine-owners, they send a sample. The argentiferous lead is assayed on a sample taken by boring into the top and bottom of both ends, and sometimes the middle of the pig. When the lot sent is large only one pig in a specified number is assayed. When the lot is small, or the pigs very rich, each pig is assayed. In some works* the sample is taken by punching five bars, driving the punch more than half through. The piercing is done diagonally across the upper side, and on the opposite diagonal on the lower. The pieces are melted in a previously-heated crucible, so as to have them rapidly melted. At a red heat the metal is stirred for five minutes, and then poured with the dross into a sample mould. The sample ought

* "Mineral Resources of the United States, 1885," p. 464.

to weigh 8 lb., and represents 10 short tons of metal. Four samples, of a little over an assay ton each, are taken from the middle of the four sides of the assay bar, cutting it through. From these four samples of an exact assay ton each are taken and cupelled. If the gold is in small quantity, 4 assay tons are taken. The results are very accurate.

The process of desilverization, as conducted in the works at Cheltenham, Salt Lake, and Mansfield Valley, consists of—

1. Softening the lead.
2. Incorporation of the zinc and separation of the zinc scum.
3. Refining the desilverized lead.
4. Treatment of the zinc scum.

The object of the desilverization, as performed in these works, is to concentrate all the silver into a very small quantity of an alloy of zinc and lead, so rich that the lead resulting from its distillation will contain 8 to 12 per cent. of silver, and to leave behind in the kettles, lead which will contain not over 5 grammes of silver to the 100 kilogrammes, and not more than 0.5 to 0.75 per cent. of zinc, and be pure enough to make white lead, and hence command the highest market price. The details of the arrangement of each of the works is different, but their general arrangement is about the same. The oldest works were arranged without reference to level, but the more recently constructed ones are built in such a way that the softening furnaces, where the metal is received to be melted and refined, are on the highest level. The desilverizing kettles are on a level below, so that the discharge can be made directly from the softening furnace into them. The liquation kettles are in a series below the desilverizing ones, and the furnaces used for liquation on the same level. The storage bins for the liquated material are generally in the distillation room, on the same level as the liquating furnaces. The refining furnaces should be on such a level that the desilverized lead can be run from the kettles into them, and from thence be cast by some simple arrangement into pigs. The cupelle furnaces are usually on the same level as the distillation furnaces.

1. *Softening the Lead.*—As the argentiferous lead comes from all sections of the country, and contains a number of impurities, in variable proportions, it must be refined or softened before it

can be desilverized. The furnace used for this purpose is called the softening furnace, in most of the works. At the Germania Works it is called the A furnace. It is a large reverberatory, with a cast or tank-iron basin, into which the hearth is built.

The object of this iron basin is to have a furnace so cool that if the lead goes down into the hearth it will chill, or if the furnace is very hot it will be caught. The larger the furnace the better. Made of cast iron, its size is limited; made of tank iron, there does not appear to be any reason why it should not be of double the size, except the uncertainty of being able to purchase the supply of lead to work continuously. With an uncertain supply, it is better to multiply furnaces, as a small amount can be better and more economically treated in a small than in a large furnace. There is a point, however, beyond which it will not be profitable to increase the size, and this will be the quantity that can be held by the kettles. The limit in the kettles evidently will be that at which a man can no longer work the kettle conveniently.

The fireplace at Cheltenham is 2 ft. 3 in. wide and 5 ft. 6 in. long. The grate is 12 in. below the bridge; the bridge is 2 ft. 2 in. below the roof, 1 ft. 6 in. above the hearth, and 2 ft. 10 in. wide. The hearth is made of a cast-iron basin which is 15 ft. 5 in. long, 9 ft. 6 in. wide in the middle, and 5 ft. 3 in. at each end, 2 ft. 4 in. deep, and 1¾ in. thick. It weighs 8 tons, and is calculated to hold 25 tons of lead. At Cheltenham, the pan forming the bottom of the furnace is cast in one piece. At the Germania Works, it is cast in three pieces and bolted together. This latter method is the cheapest; but if any of the bolts become loosened, there will be a loss of lead, to avoid which the works at Cheltenham had the pan made in one casting. At the Pennsylvania Lead Works, the pan is made of tank iron about one quarter of an inch thick, which is rivetted. The bottom of the pan rests upon rails, which are supported on brick walls on either side of the furnace. It is now proposed to water-jacket all of these furnaces, which will both reduce the quantity of repairs to be made to them, and shorten the time spent upon them. The doors of this furnace are counterpoised with pigs of lead, so that they can be very easily moved. They are bevelled and

fit into a slot, so that when they are closed and luted they are hermetically sealed.

The hearth proper is built on the iron pan bottom. It is made of firebrick laid in the form of an inverted arch, placed on a bed of coke next the pan, which is covered with a layer of brasque. If the pan is made of cast iron, the roof must not rest on it, as its pressure might increase its tendency to crack; if of wrought iron, there is no such danger. The side walls resting on it bear against projections on the rim of the sides of the pan. These precautions are necessary in all iron pan hearths, to prevent the rising of the hearth from the lead penetrating below it, and breaking it up. Notwithstanding all the precautions taken against it, this accident, which causes great inconvenience and loss, happens so often that, at the Germania Works, holes are now bored in the angles of the bottom and sides of the pan, so that the lead cannot collect. The flowing lead warns the men, before any serious accident has happened, that it is time to make repairs. These furnaces should all be placed at the highest point of the works, so that the lead and other products may descend by gravity from one furnace to the other. The hearth is made on an inverted arch, by stamping brasque on it, and cutting it out to the shape the hearth is to have. On this firebricks placed on end are put, so that the firebrick hearth has exactly the same shape as that of the brasque below. The side walls are all built with either a course of red brick or a layer of brasque behind them. The hearth inclines toward the tap-hole, which may be either at the flue end or on the side of the furnace. The tap-hole is made of cast iron; if the pan is of cast iron it is screwed on to it, if of wrought iron it is bolted on. It is lined with brasque, or arranged as described for the Mansfield refining furnace. The lead is charged through the charging door in the side of the furnace, on a spaddle with a long handle. A roller is placed in the door, to relieve the workman from the friction; the lead is thus distributed about the furnace, so as to melt readily. The charge is always melted at a low heat.

The usual charge at the Germania and Cheltenham Works is from 22 tons to 24 tons, depending on the purity of the lead. In the works of the Pennsylvania Lead Company, at Mansfield Valley, they sometimes charge as much as 25 tons to 26 tons, the charge

depending on the quantity of crasses that the lead makes. It is always made at Cheltenham so as to produce about 20 tons at the end of the operation, or a quantity sufficient to completely fill one kettle. When the furnace is hot, the whole charge melts in about two hours. It remains in the furnace from six to eighteen or even twenty-four hours, depending on the work in the kettles, which must be kept full. During this time it is kept at a low heat, and air is allowed to have free access to the surface of the metal.

The operation of softening consists in keeping the lead melted at a very low temperature, the object of which is to separate the copper by liquation, as it is much less fusible than lead. The scums containing the copper are drawn with a tool made of birchwood, so as not to contaminate the lead, as would be the case if an iron tool was used. It is always necessary to endeavour to remove all the copper, whether gold is present or not. The gases in the furnace are oxidising, and crasses containing the oxides of the foreign metal rise to the surface. At the end of three hours the temperature is raised to a dull red heat. At this temperature the volatile metals rise to the surface, are there oxidized, and mix with the lead oxides and are removed with them. The bath is kept for twelve to fifteen hours, if necessary, at the same temperature, and frequently rabbled to bring the impurities to the surface. If the lead contains from 3 to 4 per cent. of impurities, the crasses are only drawn as they form, but if more impure, a steam-jet blast is discharged directly into the bath, which, by constantly bringing fresh portions of the lead to the surface, promotes the oxidation, and the crasses are removed several times; but if the lead is moderately pure, the crasses are drawn but once, which will generally be at the end of six to seven hours. An air blast, as it acts mostly on the surface, does not answer so well as the steam, which has, however, the disdavantage that the constant wave action of the lead oxide against the surface of the brick tends to wear it away. It should be protected with a water back where steam is used. The first crasses will amount to from 1.5 to 2.5 per cent. of the charge, and are taken off at the end of from five to seven hours. Before drawing them, they are mixed with coal on top of the melted charge, to reduce any oxide of lead, and are then drawn; and if they form again they

are removed. When they no longer form, the furnace is cooled gradually, but is kept above the melting point of lead. The crasses are drawn from the working-door and are collected in a bin, where they are allowed to accumulate until there is enough to work.

When litharge commences to form, the crasses are no longer drawn, but are left in the furnace after the lead has been tapped. In refining the next charge, they give up their oxygen to more easily oxidized metals, and thus help to separate them from the lead. Quick-lime is usually added as soon as they commence to form, to keep the litharges from cutting. The time required for softening the charge varies from eighteen to thirty-six hours, depending on the amount of impurities contained in the lead; usually it is less than twenty-four hours.

Sometimes all the impurities have been removed at the end of twelve hours or less, but the charge in the furnace must stand until the desilverising kettles are ready. This is done by simply shutting the dampers, and adding only just enough fuel to the fireplace to keep the charge melted; but as all the compounds of arsenic and antimony are very fusible, the softening must be kept up as long as these form. With a charge of 26 tons, at the Pennsylvania Lead Works, from 24½ to 25½ tons of softened lead remain in the furnace.

It often happens that the charge is ready for tapping, but the desilverizing pots are in use; so that the lead is kept in the furnace at the melting point until the pots are free. It is cheaper even if the lead is extremely pure, to keep it melted in the furnace during the time necessary, rather that to cast and re-melt it.

At Cheltenham, the tap-hole opens into a deep but narrow trough lined with brasque, from which the lead is syphoned off with a Steitz syphon, Fig. 15. The brasque is made of four-fifths clay and one-fifth coke-dust. It is made as dry as it can be stamped, and is then carefully shaped and cut down to make the arch leading into the furnace. When the kettles are ready, the furnace is tapped. The tapping-spout is very large, and during the time of casting exposes a large surface to oxidation, thus increasing the losses in lead. If the furnace was sufficiently high above the pot, the lead could be tapped by a gutter directly into

the kettles. The contract is always made to have the kettles cast bottom down.

At the Germania Works, the tapping is very inconveniently done through an iron pipe, 40 ft. long and 5 in. in diameter, with holes cut into it at intervals to facilitate the removal of dross which might clog the pipe. It is necessary to heat the whole length of this pipe, to prevent the lead from chilling. This is done with coals suspended in pieces of sheet iron under it; but there must be a shield between the fire and the pipe to keep the latter from cracking.

As the softening furnace is always above the kettles, a much simpler plan is to run the lead into the kettles by gravity, using an iron trough for the purpose. At the Pennsylvania Lead Works this is accomplished with a trough made of angle iron so as to form a gutter. The other end is placed over the pot into which the metal is to be discharged. When the furnace is ready to be tapped a charcoal fire supported in a sheet-iron frame, a bracket for which is fastened to the side of the furnace, is made around the stop-cock attached to the side of the iron tank of the furnace bottom, until it is raised above the melting point of lead. The handle is then turned and the contents of the furnace discharged into the kettles. When the furnace is empty the angle iron is taken away and the space left free until the next tapping. At these works there are three of these softening furnaces, each one having three desilverizing kettles. At the Germania Works there are two, with five kettles each; at Cheltenham, one, with three kettles.

The crasses from the softening furnace are first liquated, to remove any excess of lead they may contain. At the Germania Works, this was formerly done in a reverberatory liquation furnace of peculiar construction. The hearth was 3 ft. deep; 18 in. above it a set of grate-bars was placed; the skimmings were placed on these, and the crasses remained there while the lead flowed through. This lead is very hard; it is either sold or refined with the other lead. The first crasses drawn contain most of the copper. They are always kept separate from the others. They are put through the blast furnace at the end of a campaign, with galena or pyrites, in order to concentrate the copper

in a 40 per cent. matte, which is sold. Some hard lead is produced, which is treated with the lead of the other crasses. Sometimes the crasses, without liquation, are melted in the blast furnace, producing very hard lead; this is treated with soft lead, poor in silver, in the softening furnace, and made into marketable hard lead, richer in silver, however,* than the ordinary softened lead.

At the Germania Works, a copper matte is produced which contains 20 per cent. of copper, 20 ounces to 25 ounces of silver, and a slag containing 10 ounces of silver. The matte is concentrated to 40 per cent. of copper, and is sold.

The assays of three of these concentrated samples contained—

	No. 1.	No. 2.	No. 3.
	oz.	oz.	oz.
Silver	113.54	88.	94.66
Gold	1.18	1.02	1.02

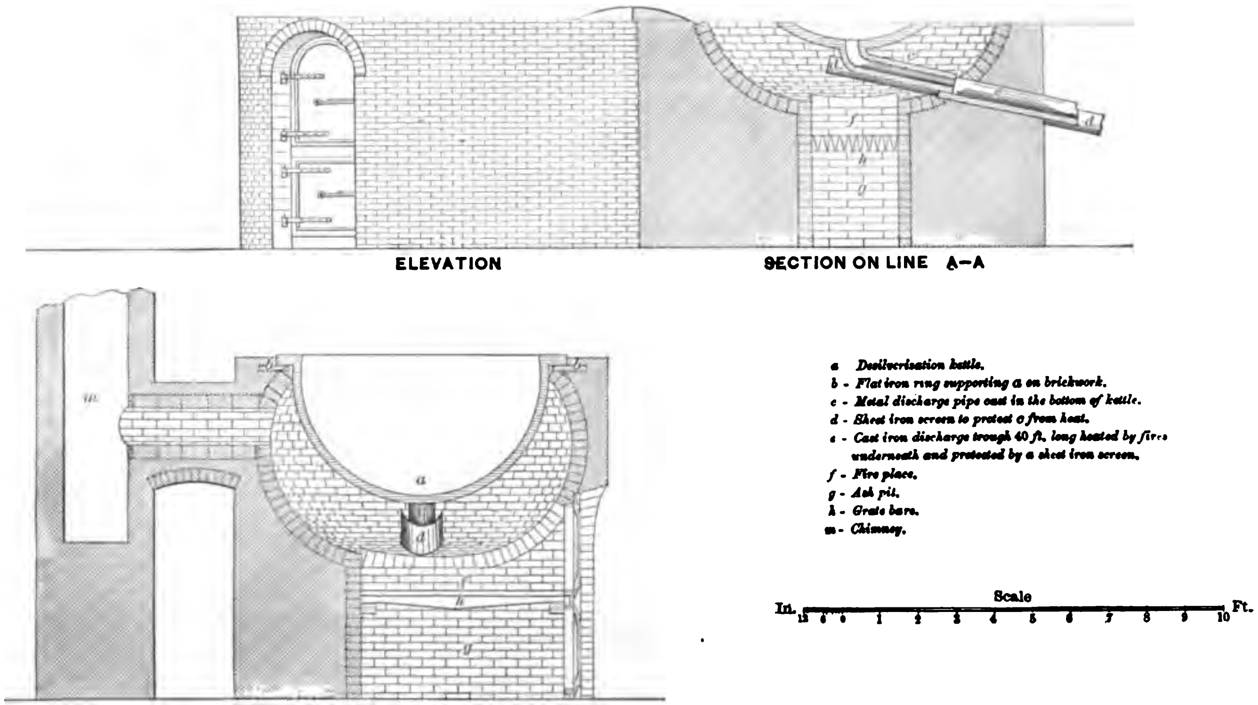
From the dust chambers connected with this furnace only a small amount of material is collected, and this very near the furnace. It contains only from 3 ounces to 4 ounces of silver. The other crasses are treated in a reverberatory furnace. The materials being at first only partially reduced, the first lead which flows carries most of the silver and is put to one side. The charge is then completely reduced. The product is a very hard lead, which is allowed to accumulate until there is enough to make a charge in the softening furnace.

If the ores contain a very large amount of antimony, there will be two or three sets of crasses after those containing copper have been removed, which will be mostly very impure litharges. The lead produced from them is a compound of arsenic and antimony, which is not refined, but sold as hard metal. The loss in lead in softening is about 2½ per cent.

2. *Incorporation of the Zinc, and Separation of the Zinc Scums.*—To effect the desilverization there are, at Cheltenham, three kettles set in a triangle, at Mansfield Valley a series of three kettles set in a row, and at the Germania Works, a series of five, set as shown in Fig. 16, the first two holding 20 tons

* "Mineral Resources of the United States, 1885," p. 472.

Fig. 16.



DESILVERIZATION KETTLES AT THE GERMANIA WORKS, FLACH'S STATION, NEAR SALT LAKE, UTAH.

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each; the next two, 7 tons, and the last, 4 tons. These kettles are set in masonry, with a fireplace underneath them. The furnace is tapped into the two upper ones alternately. The upper kettles at Mansfield hold 23 tons. The upper kettles at Cheltenham weigh 4700 lb. each, and cost between $400 and $500 each. They are 6 ft. 6 in. in diameter, and 3 ft. deep. The depth of these kettles is the same in all the works; the diameter varies with the capacity. They were first made to hold 10 tons, then 20, then 30, and now some of the works make them hold as high as 40 tons, as it was soon found that it costs but little more to treat 40 tons than it does to treat 25 tons in twenty-four hours. The limit will evidently be fixed, first by the supply of silver-lead, secondly by the number and capacity of the softening furnaces. The time that these kettles will last varies greatly. They are usually bought by weight and not on the specification of what they will do. In some of the European works they are bought by contract to last during the melting of a specified number of tons. The contractors much prefer to make the contract in this way;* but it is not usual in the United States.

At the Germania Works, the discharge spout is cast on the bottom of the kettles, and is constantly breaking. At Mans-

* I am indebted to Mr. Freudenberg, director of the silver works at Ems, for the number of kilogrammes of silver lead melted in 19 kettles made by five different foundries.

Kettle	Foundry No. I.	No. II.	No. III.	No. IV.	No. V.
	K°.	K°.	K°.	K°.	K°
No. 1	2,142,650	...	...	...	...
2	...	1,472,000	...	...	...
3	...	...	1,358,650	...	...
4	...	501,000	...	...	...
5	...	...	...	1,368,510	...
6	368,290	...	...	...	...
7	862,225	...	...	...	...
8	1,618,550	...	...	...	...
9	...	...	990,600	...	...
10	...	...	1,189,975	...	...
11	...	...	543,500	...	...
12	...	...	6,296,800	...	...
13	...	977,000	...	...	...
14	...	597,500	...	...	...
15	...	...	924,000	...	...
16	...	...	596,500	...	...
17	...	...	...	...	622,500
18	...	...	...	...	974,250
19	...	...	...	...	1,063,125

field, the middle one has a spout at the bottom, which communicates with the third and smallest. At Cheltenham the kettles are discharged by the Steitz syphon, which is a much simpler method, and much less expensive than having pipes cast on the bottom, as they always increase both the cost of the kettles and the liability of loss of lead. The kettles are filled with melted lead from the softening furnaces. When they are full, they are heated up to the melting-point of zinc, which takes about one hour. It is important that the heat should be high enough to melt the zinc readily. The kettle is so large that there is but little danger of over-heating. When the temperature is at the right point, the zinc is added. At the Germania Works and at Mansfield, the zinc is thrown in or laid on the top of the lead, and incorporated as it melts. At Cheltenham, it is placed in an iron cage, which is let down to the bottom of the pot. The amount of zinc to be added will generally be about one pound for every 5½ ounces of silver. This will usually amount to between 250 lb. and 550 lb. to each kettle. In general, with ores varying from 100 to 300 ounces of silver, 1.4 to 3 per cent. of zinc is added. It is not all added at once, but sometimes in two and sometimes in three additions, the proportions being determined by assay in each case. These additions should be so regulated as to make the richest possible alloy at first, in order to shorten the process as much as practicable, to diminish the liability to oxidation when it is liquated, and to produce a lead containing not more than 0.1 to 0.2 of an ounce of silver.

At Mansfield, the lead contains from 50 to 400 ounces of silver. To this, from one and one-tenth to two per cent. of zinc is added, in four additions. The zinc is thrown in on the top of the melted lead, and then is stirred into it by a tool, 5 in. by 10 in., with a long handle. After the first addition, it is stirred for half an hour. The scum is then allowed to rise and cool, until there is a ring of 3¼ in. around the outside. It is then skimmed with a perforated skimmer until the lead is bright. The other additions are made in the same way.

At the Germania Works, for a charge containing 60 ounces of silver and ⅓ ounce of gold, 1.85 per cent. of zinc was added. For a charge containing 140 ounces of silver, and 3.8 ounces of gold,

2.3 per cent of zinc was used. Of this 0.5 was added in the first addition, 0.4 in the second, and 0.1 in the third. For a charge containing 350 ounces, 2.6 per cent. of zinc was used.

The following Table, prepared by Mr. A. V. Weisse, of the Germania Works, gives the amount of zinc used in two charges:

Example.	Total Weight of Softened Lead.	Silver contained in Grammes to the 1000 Kilos.	Gold contained in Grammes to the 1000 Kilos.	Zinc Used.
	lb.			per cent.
No. 1 . .	402.442	4300.	125.	2.3
No. 2 . .	402.224	4256.7	127.45	2.6

To be sure of lead at 5 grammes from lead containing 1000 to 1400 ounces of silver, at least $1\frac{1}{2}$ per cent. of zinc must be added. Pure zinc is no longer used for all these additions. The second, third, and fourth scums of a previous operation, which are not very rich in silver, are used for the first and sometimes for the second addition, thus greatly reducing the amount of zinc required for the operation. When the lead is very poor in silver, the first addition is used several times, in order to make it as rich as possible. The object of dividing the additions is to arrive, as quickly as may be, at the highest percentage of silver, and to get an alloy so rich that there will be little liability to loss in the subsequent liquation, thus shortening and cheapening the process. The amount to be added in the first charge will depend on the amount of copper in the lead. If it contains but a small amount of copper and some gold, 100 lb. are added, at Cheltenham. If there is much copper, more zinc must be added to bring out the copper, as most of the copper comes off with the first crasses. If gold is present in large proportion, the quantity of zinc must be increased, since all the gold comes off with the first scums. In making the assay, to ascertain whether the gold has been removed, small quantities should not be taken, for with them no traces of gold will be found. It has been ascertained by experience, that it is best always to take from 8 to 10 assay tons, in order to get a

weighable amount.* It is a question whether, as the gold has a stronger affinity for zinc than it has for silver, it would not be better to make a doré bullion alone, by taking out the gold and silver by two or three additions of zinc, rather than make doré bullion and fine silver by repeating the additions of zinc. If no gold is present, two-thirds of the charge of zinc necessary for the whole operation may be added in the first charge; it is then stirred from one-half to three-quarters of an hour with a flat spatula, which is 17 in. in diameter, attached to a piece of gas-tubing 6 ft. long. The temperature during this time is kept above the melting-point of zinc. The tool is made to work from the sides toward the centre, with a downward motion at the same time. In order that the workman shall not have to bear the whole weight of the stirrer, it is suspended on a hook attached to a chain hung in the ceiling. The stirring is very difficult work, and a great many tools have been devised to do it mechanically, with no great success. Dry steam from a ¾-in. pipe, placed 12 in. to 15 in. below the top of the melted lead, in the centre of the kettle, does this stirring very thoroughly; it does not require much attention, and by its use for from half to three-quarters of an hour much labour is saved. The lead is also made much poorer in silver than can be done by hand work. Any metal that adheres to the sides of the kettle must be detached with a chisel-shaped tool and thrown to the centre of the kettle. When the zinc is thoroughly incorporated, the fire is drawn, and the kettle allowed to cool until the zinc alloy, which contains the silver, rises and floats on the top of the melted lead. This time depends on the heat of the metal, and on the season of the year. In summer, it is four hours; in winter, only two.

It has been proposed† to introduce water running through pipes bent to the shape of the kettle, for the purpose of hastening the cooling the bath. These are movable so as to prevent the metal from cooling on them. The movement of the pipes reduces the temperature about equally throughout the bath. Usually the upper part cools faster than the lower. By this method it was expected to materially reduce the quantity of zinc required as well as the

* "Mineral Resources of the United States, 1885," p. 467.

† "Mining and Scientific Press," San Francisco, vol. xliv., No. 5, 1882.

time of the operation, but it has not worked very satisfactorily. Generally, however, the cooling is done by either drawing or banking the fire, or by throwing water on the surface of the metal, and taking off the consolidated crust as quickly as it forms.

The skimmings are taken off in perforated ladles suspended on chains, and put into one of the smaller kettles. These first skimmings are carefully separated from the rest, if the lead contains either much gold or much copper, or both. At Cheltenham, the skimmings from the first addition of zinc are charged into a small kettle between the two large ones. At the Germania Works, kettles Nos. 1 and 2 are skimmed into Nos. 3 and 4. If the skimmings come from the first addition of zinc, they are partially liquated in Nos. 3 and 4, and transferred to No. 5, where the liquation is completed. All the lead in Nos. 3 and 4 is then put back into Nos. 1 and 2, ready to receive the second addition of zinc. The skimmings from the 2nd, 3rd, and 4th additions of zinc are not liquated, but are used over again. The amount of labour required is one man to each kettle. The kettle is left until it is full, and is then fired up and partially liquated, which takes about an hour. The kettle must not be heated too hot in this liquation, for there would be danger of oxidising the zinc, in which case the silver would go back to the lead. The lead separated in liquation is put back into the large kettle, No. 1, before the second addition of zinc.

At Mansfield, all the skimmings except the first, which contains copper and may contain gold, are ladled into the middle kettle, which is kept heated, and are liquated at once, the lead flowing into No. 3. The lead which collects there is put back into No. 1 with the next charge of lead. At Cheltenham, the zinc skimmings are taken from kettle No. 1, and liquated in No. 2. While the second addition of zinc is being made, the liquated lead is removed to No. 3. The six tons in No. 3 are put back into No. 1, after the second addition of zinc.

The lead remaining in the kettle after the first skimming should not contain more than 20 ounces. The zinc for the second and third skimmings is not liquated, but used in the next operations. The skimming is made into the adjacent kettle. After making an assay of the melted lead, to ascertain

what is required, the next addition of zinc is made, and the skimming continued about the same time. After the second skimming, there should not be more that 10 ounces to 15 ounces of silver remaining. An addition is made, if the assay shows it to be necessary. The last two charges are placed partly on top of the melted lead and partly in the cage. It is then stirred for three-quarters of an hour and left to cool down. The skimmings are liquated as before. The lead contains from one to one and a half ounces of silver. A new addition of zinc of about 100 lb. is made; after which the lead will not contain more than two to three-tenths of an ounce of silver.

At Cheltenham, there is not more than one-sixth of an ounce of silver remaining when the lead is tapped into the refining furnace. Frequently, the last skimmings are too poor in silver to admit of treating. They are put to one side, and form either a part or the whole of thc first additions of zinc in the next kettle.

It has been suggested* that the assays taken from the kettle to determine the value of the metal are often taken either too soon or when the metal is too hot. In the latter case it is said that the silver rises to the top and gives a higher value to the metal than the average of the kettle.

At Mansfield, poor lead is not tapped if it contains more than one-tenth of an ounce of silver to the ton, and the merchant pig assays 0.075 ounce to 0.15 ounce.

When the Germania Works were first built, the Flack process was used. The liquated zinc skimmings were charged in a blast furnace with a very basic slag, and small pressure of blast. The result was rich lead, and a rich slag. In the condensation chambers, a very impure oxide of zinc was collected, which was but a small part of that actually charged in the furnace. As the use of this process occasioned a loss of from $18,000 to $25,000 a year in zinc, it was abandoned, and the Faber du Faur furnace was introduced in its place.

It is always best to use good zinc for the separation. An attempt was made at the Chicago Silver Smelting and Refining Works, to economise in this direction by using scrap zinc;

* "Mining and Engineering Journal," vol. xxxvi:, p. 274.

but it was found that the lead, after its use, sometimes contained as high as 18 ounces to the ton, and the attempt had to be abandoned.

The following statement of several charges at the Germania Works is made by the superintendent, Mr. A. V. Weisse :*

—	No. 1.	No. 2.
Number of pounds charged in the softening furnace	41,614	40,120
„ grammes of silver	5,700	1,980
„ „ gold	110	10
First addition of zinc, from second and third additions of a previous operation, in pounds	4,000	3,000
†Grammes of silver in lead after first addition	1,360	1,600
Second addition of zinc in pounds	600	600
†Grammes of silver in lead after the second addition	20	30
Third addition of zinc in pounds	80	125
†Grammes of silver in lead after the third addition	trace.	6

The following Table was prepared by Mr. E. F. Eurich, of the Pennsylvania Lead Company :

Desilverization.	No. 1.‡	No. 2.
Quantity of work lead charged in the kettle	87,294 lb.	
Taken off; "Schlicker" (cuprous oxide)	3,497 „	
Pure work lead	83,797 „	62,895 lb.
Silver contained	6,305.6 oz.	6,165.9 oz.
Quantity of zinc added	1,760 lb.	1,260 lb.
Weight of skimmings after liquation	9,525 „	6,362 „
"Abstrick" from dezincation of poor lead	7,810 „	3,500 „
Oxides and metallic lead from the market kettle	1,000 „	700 „
Lead from liquation of zinc-crust	808 „	
Market lead	67,104 „	53,420 „

At Cheltenham, the liquated skimmings, still soft, are thrown on iron gratings from 1 in. to 1½ in. apart, and pushed through in order to reduce it to pieces of small size, which can be more conveniently introduced into the retort. In most of the works, it is thrown upon an iron plate in front of the kettle, and in

* Mining Commissioners' Report for 1875.

† The grammes are given per 1000 kilogrammes.

‡ No. 1 is lead taken directly from the shaft furnace, which has not been softened. No. 2 is softened lead.

order to break it up, is rapidly moved about with a rake, and if necessary cut up with a shovel, so that the pieces are about the size of a hickory nut.

That it is very necessary to refine the lead before adding the zinc is shown by the following experiments made by Mr. C. Kirchhof.* At the Delaware Lead Works 20 tons of lead, containing 95.5 per cent. of lead, and containing antimony, arsenic, zinc, bismuth, and copper, were slowly melted in a kettle. During the melting it was carefully crassed to remove as much copper as possible. At the same time a charge of refined lead was treated. The zinc was added as usual. The Table below gives the result.

	I. Not Refined.			II. Refined.		
—	Silver per Ton in Ounces.	Charge of Zinc.		Silver per Ton in Ounces.	Charge of Zinc.	
	oz.	No.	lb.	oz.	No.	lb.
Before adding zinc . .	85.60	...	...	94.90		
After 1st charge . .	85.50	1	250	85.60	1	150
,, 2nd ,, . .	85.30	2	250	47.60	2	150
,, 3rd ,, . .	83.80	3	150	16.10	3	150
,, 4th ,, . .	83.50	4	100	1.70	4	150
,, 5th ,, . .	83.00	5	100	.18	5	100
,, 6th ,, . .	48.20	6	100			
,, 7th ,, . .	8.20	7	100			
,, 8th ,, . .	.80	8	70			
,, 9th ,, . .	.15	9	30			
Total . .	...	...	1150	...	...	700

The two operations were carried out under exactly similar circumstances. The amount of zinc used with the unrefined lead was 2.87 per cent. with nearly double the amount of time and labour required for the second. The zinc used for the refined lead was 1.75 per cent. With the unrefined lead, the amount of marketable lead produced was 43 per cent., with the refined, 72 per cent. The unrefined produced a very large amount of impure zinc scum, which increased the time, cost, and losses in distilling, cupelling, and working the products.

3. *Refining the Desilverized Lead.*—The lead in kettle No. 1,

* "Metallurgical Review," vol. i., p. 224.

Fig. 17

LEAD REFINING FURNACE,

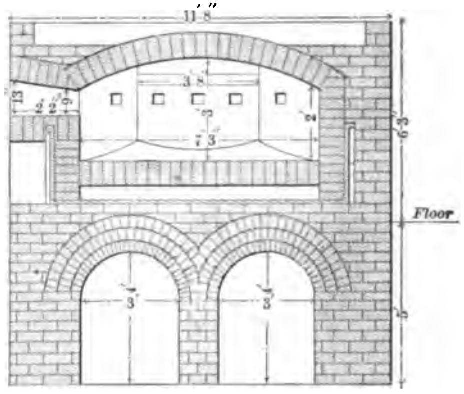
AND MARKET KETTLE.

Holding from 18 to 19 tons.

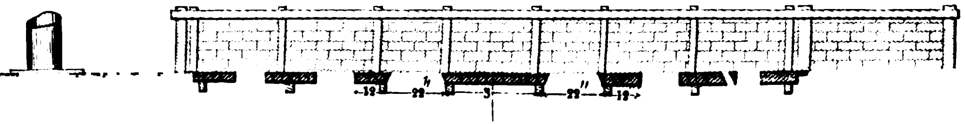
AT GERMANIA WORKS, UTAH.

SCALE

In. 12 6 0 1 2 3 4 5 6 7 8 9 10 Ft.



SECTION THROUGH C.--D.



PLAN.

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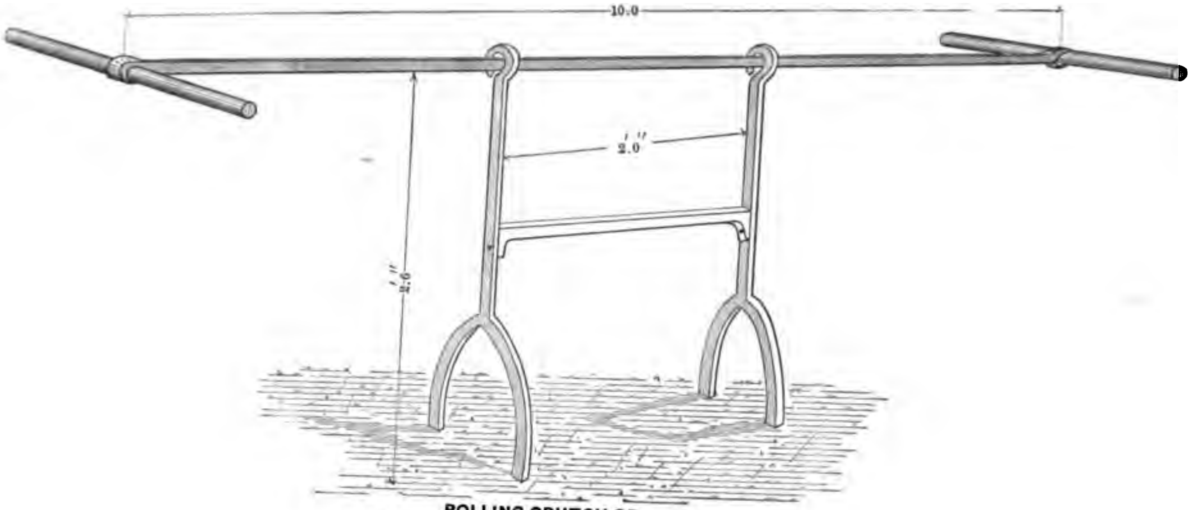
which contains usually ¾ per cent. of zinc, no matter what the heat is, or how much zinc is added. At the Germania Works, it contains from 0.7 to 1 per cent. of zinc and antimony together. It must be refined, to separate the zinc and get it ready for the market. This operation is one of refining; but in the West it is known under the name of "calcination." This is done in a furnace with a cast or tank-iron bottom, like the softening furnace, holding from 20 to 40 tons. At Mansfield Valley, the bottom is made of tank iron, with a pipe with a stop-cock attached to the side, for discharging it. Fig. 17 represents the furnace used at the Germania Works. The one used at Cheltenham is essentially the same; it is a little larger, but the dimensions vary only a few inches. The fireplace is 2 ft. 3 in. wide and 4 ft. 5 in. long. The bridge is 8 in. below the roof on the fireplace, and 11 in. on the hearth side. It is 2 ft. 10 in. wide, 3 ft. 6 in. long, and 2 ft. above the hearth. The hearth is 13 ft. 4 in. long, and 7 ft. 3 in. wide in the middle, and 3 ft. 6 in. wide, both at the fire-bridge and the flue. It is here made of one casting; at the Germania Works it is cast in three pieces, as shown in the section A-B, Fig. 17. The arch is 2 ft. 9 in. above the floor of the laboratory. It has three openings, 4 in. square in the fire-bridge, and two on its side, for the introduction of air. The charge remains in this furnace from eighteen to twenty-four hours. The surface is constantly exposed to the air entering the furnace by the air-holes at the bridge. Towards the end of the first half of the time that the charge is to remain in the furnace, the bath is skimmed. By this time all of the zinc will have been either volatilized and carried off in the gas, or have been oxidized and be contained in the scums which have been withdrawn. After this the antimony is oxidized; some of the lead oxidizing at the same time, the progress of the operation is ascertained, by making small test bars from time to time, on which the quality of the lead is determined. As soon as the antimony is all gone, the lead is fit for corroding, and is cast to be sent to the white lead works. The skimmings amount to from 1 ton to 1½ tons. They contain from 45 per cent. to 50 per cent. of lead, and most of the zinc and other remaining impurities. The charge is rabbled, after the oxides have been removed, but any others

which form are allowed to remain until the furnace is tapped into the polling kettle, which is usually about twenty hours after the charge is made, and are then polled.

At Mansfield Valley the refining is done in twelve hours. The lead is not polled, but is cast into pigs directly from the furnace. At Cheltenham and elsewhere, where the lead is cast before all the antimony is out, it is polled. The polling-kettle is placed at the flue end of the furnace. The lead flows into a deep cast-iron channel lined with brasque, from which it is syphoned off. The top of the kettle is about 6 ft. from the floor. Directly in front of the kettle, and about 2 ft. below the floor level, there is a sunken track upon which a car is run, the top of which comes up to the level of the floor. The car is about 6 ft. wide, and receives the pigs and carries them to the store-house. There is a space of 4 ft. between the car and the furnace. The polling is done in eight hours. The wood is held at the bottom of the kettle by a crutch, Fig. 18. The same apparatus is used at the Germania Works, except that instead of the crutch, the bars are straight and pointed, the holes are bored in the wood to receive them. Short sticks of green wood are used, but to insure a plentiful escape of steam, all the wood for this purpose is kept soaking in a pool of water. Three or exceptionally four pollings are made, the number depending on the quality of the lead; each polling lasts about an hour, so that the furnace is ready to receive a new charge as soon as the one refined in the softening furnace is desilverized. There is a great advantage, both in the reverberatory furnace and in the polling-kettle, in using dry steam, both to oxidize the zinc and to bring fresh surfaces in contact with the air. It is for this reason that the very wet wood is used in the polling kettles, as it is the steam which does the work, producing a very high quality of lead. The advantage of polling with wood in the reverberatory furnace has long been recognised, but it is only lately that steam, which is much more manageable, has been used. When steam is used in polling, it is introduced from an iron pipe 1½ in. in diameter, which is let down nearly to the bottom of the kettle. The steam has a pressure a little above what is necessary to overcome the plumbostatic pressure. The current of steam is kept up for from

Fig. 18.



POLLING CRUTCH OR
WOOD-HOLDER FOR THE POLLING POTS

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ten to sixty minutes, the time depending on the purity of the lead. Generally, however, in the United States, the lead contains so large an amount of antimony, that the softening furnaces must be used. The polling kettle is only used as an auxiliary, so as not to delay the preceding operations, in case the antimony is not all out when the next charge is ready to go into the softening furnace. It is much better to poll with steam, than to keep the lead longer in the softening furnace, and thus keep back the rest of the process. The weight of the dross collected from a kettle at the Germania Works, which was polled four times, is given below.

	lb.
1st polling	1301
2nd ,, ...	881
3rd ,, ...	671
4th ,, ...	290
Total	3143

The crasses from all the pollings, usually amounting to from 1000 lb. to 2100 lb., are melted, at the Germania Works, in a reverberatory furnace, and make common soft lead. The crasses from the softening furnace, however, make silver lead, which is treated by zinc. Those from refining, which at the Germania Works is called calcination, make soft lead of ordinary quality.

The following Table gives the quantity of skimmings for examples Nos. 1 and 2, page 93.

	lb.
From refining furnace	31,700
Polling kettle	20,352

Quantity of work lead taken from the polling kettle, 76.25 per cent.
Silver contained in the market lead, per 1000 kilogrammes, 6 grammes.

The polling kettles at Cheltenham, are emptied by the Steitz syphon, Fig. 15. To do this, it is first heated in the melted lead; it is then turned over so that the funnel-end, *a*, is uppermost. The stop-cock, *b*, is then opened, and melted lead poured into the funnel, *d*, which is inserted into *a* for the purpose. When full, the stop-cock, *b*, is closed, and the syphon, full of lead, is then turned over into the kettle, and placed in the position shown in Fig. 15. The joints of this syphon are made of gas-pipe fittings. At first it was supposed to be necessary to make them perfectly air-tight, but afterwards it was found that

when six or eight threads of the screw were run into the fitting, the joint was lead-tight, and perfectly flexible. The end of the syphon, where it turns down to discharge the lead,

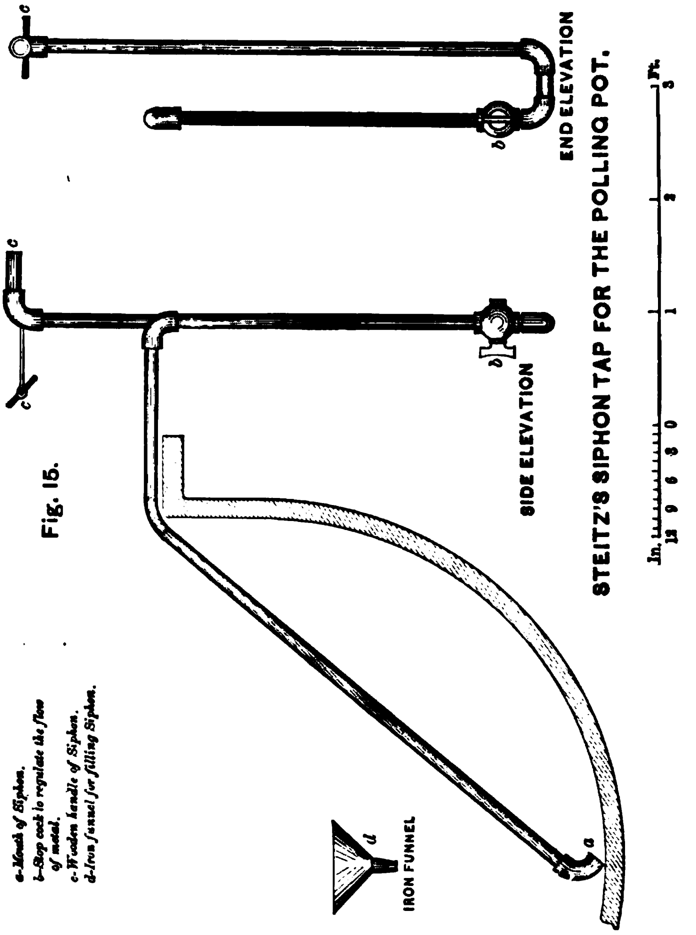


Fig. 15.

STEITZ'S SIPHON TAP FOR THE POLLING POT.

is a simple gas-pipe fitting, to which a handle, *c*, is attached for convenience of moving. While the lead is not being cast, the vertical arm is simply turned up. When the car with the pig-moulds is ready, the syphon is turned down, being held by the handle, and is moved from one pig-mould to the other in succession, as they are filled with lead. The joint is long enough to allow of filling all the moulds without moving the car.

At the Pennsylvania Lead Works, the same arrangement for

discharging the furnaces is made as has been described for the softening furnace, except that the angle iron at the end away from the furnace is supported above a swinging trough suspended to the ceiling one end of which is circular in shape. The tap-hole of the furnace is opened; the lead flows in the angle-iron launder to the foot of the swinging trough, the spout of which can be directed at will toward any ingot mould which is to be filled. When the casting is complete the angle iron is taken away, and the swinging trough removed. As this involves the use of a large number of ingot moulds in other parts of the same works, the casting is done in pig moulds attached to bogies, of which there are four. The first is filled and rolled away to cool, the second and third are filled and rolled away in the same way. When the third is withdrawn, and by the time the fourth is ready to fill, the first is cool enough to be tipped, and is brought to its place to be filled again by the time that the fourth commences to fill. This method requires but few moulds, but needs quick and dexterous men.

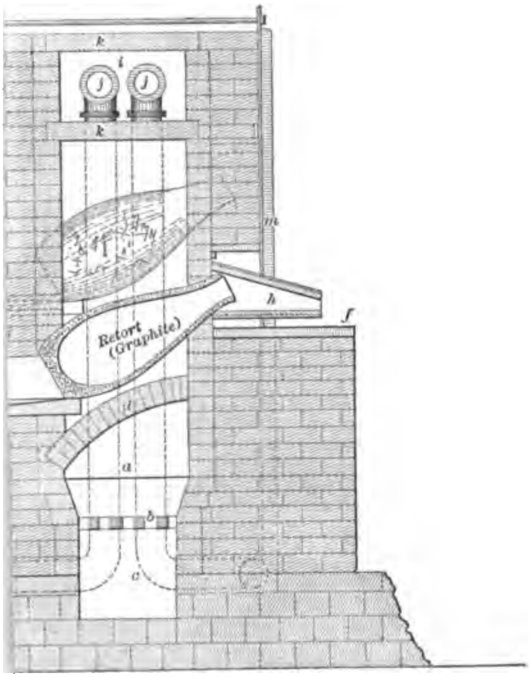
4. *Treatment of Zinc Scums.*—The zinc crusts from the liquation, are reduced to small pieces and distilled. If it has been carefully liquated, it will not contain more than 4 to 6 per cent. of lead. The distillation is done in graphite retorts in fixed furnaces, as was formerly the case at Bloomfield and Cheltenham, or in Faber du Faur's tilting furnace.

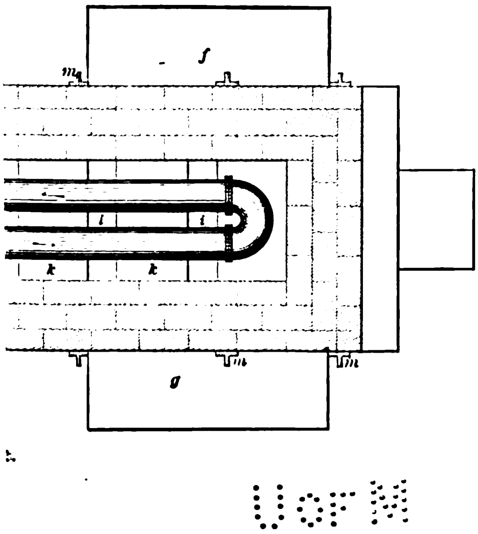
At the Germania Works, the Flack process is sometimes used for commercial reasons, but it was never considered good metallurgy. It consists in charging the zinc scums in a shaft-furnace with the drosses from refining and ores of all kinds. The result of this treatment is a rich silver lead, but the greater part of the zinc is lost. From a metallurgical standpoint, this treatment is very objectionable, and should not be imitated; but the commercial conditions in Utah are so peculiar that it has proved financially successful, owing probably to the great skill with which the process is managed; for a bad process well conducted may sometimes be made successful. In almost every other establishment in the country, the zinc scums are retorted. The retorts used at Bloomfield, N. J., Philadelphia, Cheltenham, and the Germania Works, are shown in Figs. 19, 20,

and 22. They formerly varied but little in different works. They were made of as small a capacity as 200 lb., but this was found to be too small. They have been recently made as high as 700 lb., but this is rather large. The usual capacity is between 400 lb. and 500 lb. Generally they are ¾ in. thick on the sides and nearly twice that on the bottom; the neck is 7 in. long and the body of the retort is 2 ft. The diameter at the extremity of the neck is 5½ in., but where it joins the body it is 8 in. The body in its widest part is 14 in., but it is only 9 in. at the end. These retorts are made of New Jersey clay and chamotte with 25 per cent. of graphite. They were formerly one of the largest items of cost in the conduct of the operation.

One of the first furnaces used for the distillation of the zinc was invented by Mr. W. M. Brodie, and has been constructed in several works. It consists of a large chamber, in which six retorts are placed in two levels, as shown in Fig. 19. These are heated by a fireplace, 2 ft. 10 in. long and 15 in. wide, with cast-iron grate-bars which is blown by a forced blast which enters the ash-pit at *c*, having first been heated in two hot-air pipes which are placed in compartments above and behind the furnace. The retorts are protected from the direct action of the fire by the arches, *d*. The heat escapes by the flues above the retort chamber, passes into the chamber above, down at the back, and out of the furnace by an underground flue. The retorts are the ordinary graphite retorts, holding from 450 lb. to 500 lb., so that the furnace would hold from 2600 lb. to 3000 lb. of alloy at a time. Each retort has a condenser, *h*, attached to it, and in front of it a charging-table, *f*, covered with cast iron. It is necessary to remove the condenser, as in the other furnaces, to clean the retort. The furnace is tapped on the back side at *e*, from holes ¼ in. in diameter, bored through the bottom of the retort, into moulds placed on the iron ledge, *g*.

If the material charged is pure, the time required for an operation is twelve hours. If it is not, it may require as much as twenty-four hours, depending on the quality of the material charged. One man does the work of the six retorts. The amount of fuel required is one ton of coal for one ton of alloy. The results do not differ materially from those of the other furnaces,


k
i
j
j
k
m
h
f
Retort
(Graphite)
d
a
b
c


f
m
i
i
k
k
m
m
g

except that the operation is longer. They were constructed in the now abandoned works at Bloomfield, N. J., and the works of Messrs. Tatham, in Philadelphia.

The following Table of the results of the working of this furnace has been prepared for me by Mr. C. Kirchoff, Jun., who had charge of these furnaces while they were working:

TABLE OF CHARGES IN THE BRODIE FURNACE.

—	No. of Shifts of 12 Hours.	Pounds of Zinc Scum Liquated in Kettle.	Pounds of Bituminous Coal used for Distillation.	Number of Barrows of Charcoal.*	Yield in Rich Lead.	No. of Charges.
No. 1† . .	19	9,916	22,000	6	8,681	65
,, 2† . .	17	13,656	26,000	6	10,862	66
,, 3† . .	21	19,944	14,000§ with 1 ton of coke.	3	14,511	60
,, 4 . .	26	19,622‡	34,000§	...	19,015	73
,, 5 . .	26	27,324‡	20,000	...	23,738	73
,, 6 . .	28	21,114‡	...	...	11,927	
With hot air .	28	17,300‡	...	...	14,902	83‖

The following Table covers five runs: unfortunately the lists do not specify how many retorts were fit for further service at the end of a run:

No of retorts . .	I.	II.	III.	IV.	V.	VI.
1st run . . .	13	7	11	12	9	11
2nd ,, . . .	12	10	9	10	9	12
3rd ,, . . .	10	13	8	10	12	7
4th ,, . . .	12	15	12	12	15	7
5th ,, . . .	13	14	7	13	13	13

The figures give the number of charges made in each retort.

At Cheltenham the retorts are set in the furnace, Fig. 20, with the level of the bottom below the mouth, and so inclined that the

* A barrow contains about four bushels.

† Nos. 1, 2, and 3 yielded 4049 lb. of zinc regained

‡ Mixed with copper scum. First scum at first kept separate, but not afterwards.

§ One-half anthracite and one-half bituminous.

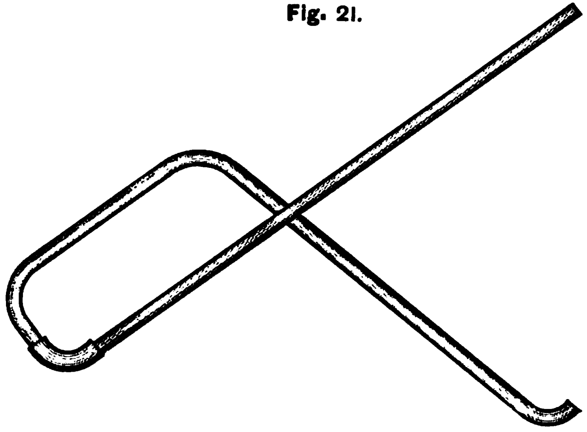
‖ Four retorts were still good.

syphon, Fig. 21, can draw out nearly the whole of the silver-lead. Some of it will remain, but this is no disadvantage, as it is not lost. It is collected when the retort is broken. Its presence, however, requires that a reducing temperature should always be kept up in the retort, otherwise litharge would form and the retort be quickly pierced. The furnace is a cube of firebrick, 3 ft. in cube, braced in every direction with wrought-iron bands 3 in. wide. On the top there is a round hole, *h*, 10 in. in diameter, for the introduction of the fuel; on the front, is an opening for the neck of the retort, *c*, and on the back, a square flue, *g*, leading to the chimney. The retort is introduced from the bottom. The furnace has twelve grate-bars 1 in. square, and is supported in front on masonry, *b*, built with two steps, each of which is 18 in. high, but vertical behind. The retort is supported on a pillar of brickwork, *d*, resting on the ground, through which the grate-bars pass. It is thus in the centre of the furnace and is surrounded on all sides by fuel. It costs from $14 to $16 and lasts from 15 to 30 turns. When it breaks, it is not because it is worn out, but because the workmen break it in trying to force off the cinders attached to it. Five of these funaces were arranged in a house by themselves, about a hexagonal chimney, and connected with it by the flue, *g*, 3 ft. long. The sixth side of the chimney is occupied by a melting furnace. Only three of the furnaces are run at a time, the others being kept in reserve in case of accident or necessary repairs.

The fuel used was at first coke, which was given up because the clinkers attached themselves to the retorts. In trying to remove them, the men constantly broke the retorts by poking them, while the cinder was soft, with iron tools, through the opening for the introduction of fuel. Petroleum was then used with great success, but the furnaces were finally abandoned at these works for Faber du Faur's furnace.

The charge of 380 lb. of zinc skimmings is introduced with a spoon, immediately after the preceding operation is finished. Two small scoopfuls of small charcoal are added at the same time. The heat is so high that most of the charge melts at once. A prolong *e*, Fig. 20, 2 ft. long, 4 in. in diameter at the small, and

Fig. 21.



STEITZ'S SIPHON TAP FOR THE DISTILLATION FURNACE

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9 in. at the large end, is then put on and luted. It is partially filled with charcoal. The prolong is covered on the outside with sheet iron, to protect it against accident. It is supported on a cast-iron shelf, *f*, which can be raised or lowered at will by detaching a bar underneath it. This is necessary to prevent the weight of the prolong breaking the retort while the furnace is working. When the charge is drawn, it must be let down so as not to interfere with the syphon, Fig. 21.

The zinc commences to distil in about three-quarters of an hour. Metallic zinc collects in the condenser. Some blue powder and oxide of zinc also form there. The object of the charcoal is to prevent the formation of oxide as much as possible. The zinc is allowed to accumulate, and is drawn from time to time with a spoon into a mould placed in front of the prolong. About 60 per cent. of the zinc is recovered as metallic zinc and is cast into slabs to be used over again ; 20 per cent. is recovered as blue powder mixed with oxide, the rest of the zinc is lost. The amount of silver contained in what is recovered is not appreciable. When the zinc is nearly distilled, a small piece of wood is put into the retort to make a reducing atmosphere, to prevent the formation of litharge, which would pierce the sides, and to form a current of gas from the inside to the outside of the retort. The charge of rich silver-lead remaining after the zinc is distilled is drawn with the iron syphon, Fig. 21. It must be heated before it is introduced, and is handled with heavy mittens. The lead is cast into pigs ready for cupellation.

Before the invention of the Steitz syphon, the neck of the retort, which was necessarily built into the masonry of the furnace, had to be disengaged while it was at a white heat, before the rich silver-lead could be discharged from the furnace. The percentage of breakage was thus greatly increased, so that between the necessity of getting rid of the clinkers on the outside of the retort, and the necessity of disengaging the neck every time it was discharged, the number of retorts broken was very large. The syphon proved to be a complete remedy, but was difficult to use, much more so than the polling-pot syphons. The objections to using these furnaces was not only the breakage of the retorts but the large quantity of fuel they consumed.

The Brodie furnace, with two tiers of retorts, consumed less than the Cheltenham furnace, but the retorts were more difficult to manage. The use of petroleum seemed to be a real progress, and the use of gas was proposed, when the invention of the tilting-furnace overcame all difficulties, and it is now almost universally used for this purpose.

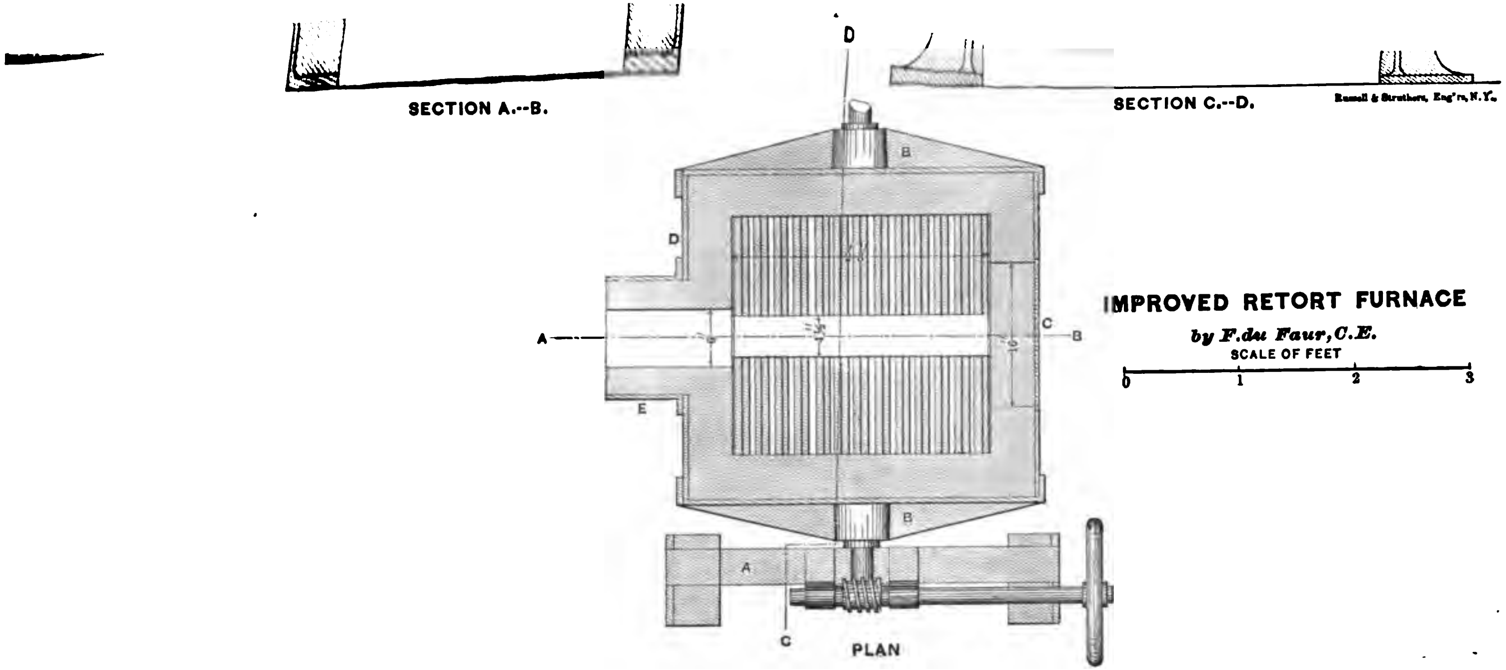
The general shape of Faber du Faur's furnace is essentially the same as that at Cheltenham ; but it is suspended on pivots, so that it is capable of rotation by means of a worm attached to a handwheel, as in the American type of the furnace, Fig. 22, or by means of a lever, as in the German type, used in Newark and in Prussia, Fig. 23. The furnace is 3 ft. 3 in. by 2 ft. 11 in. in section, by 3 ft. high on the outside, 2 ft. 1 in. by 2 ft. 3 in., and 2 ft. 9 in. from the grate-bars to the centre of the arch on the inside. There is an opening 11 in. in diameter on the top, for the introduction of the fuel, and on the back a flue 6 ft. 6 in. leading to the chimney. There are 12 grate-bars 1 in. square and 2 ft. 9 in. long set on edge. The retort is built into the furnace in the same way as at Cheltenham.

Fig. 24 gives the plan of the furnaces at Salt Lake, showing the disposition of the eight furnaces, *a*, with regard to the main chimney, *g*, and a section across the flue, *f*. At Mansfield Valley, the chimney is at the end of the line of furnaces. The weight of the iron for a furnace is nearly as follows :

	lb.
Cast-iron box	1260
Grate-bar bearers	306
Two standards	530
Cast iron	2096
Wrought-iron bars	181

The wrought iron costs from $150 to $165.

The furnace is fired until the retort gradually arrives at a dull red heat, when a charge of 250 lb. to 400 lb. of the alloy, broken up while still soft, in order to get it of a suitable size for the charge, and mixed with 5 lb. to 6 lb. of small charcoal, is introduced with a scoop. It is brought to the retorts at Mansfield in a box on wheels, about 3 ft. by 3 ft., and a little lower than the mouth of the retort. As soon as the retort is charged, the



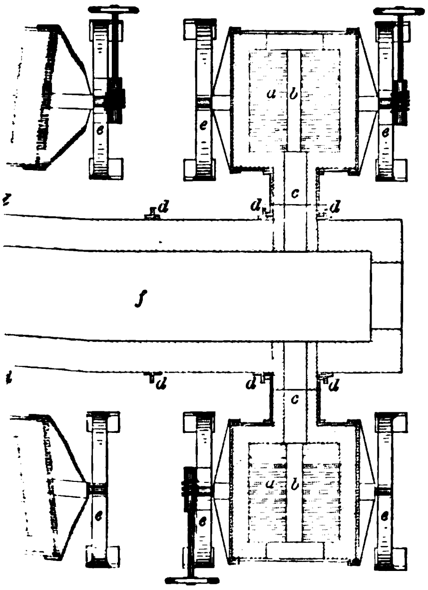
IMPROVED RETORT FURNACE

by F. du Faur, C.E.

To face page 104

- *Faber Du Faur's tilting furnace.*
- *Support for retort.*
- *Flue jointed so that the furnace may swing.*
- *Vertical T iron ties holding brick walls together.*
- *Stands supporting furnaces*
- *Main flue.*
- *Chimney 60 feet high.*

1 2 3 4 5 6 7 8 9 10 Ft.



TATION, NEAR SALT LAKE, UTAH.

temperature is gradually raised to a white heat, and when the zinc vapours begin to appear, the condenser, made in the same way as that at Cheltenham, is put on. At Mansfield, they use for a condenser a retort, No. 100, with the bottom broken out, and a hole punched in the side to discharge the zinc. A piece of common stove-pipe is attached to the mouth to carry off the gases.

The retorts usually last fifteen to twenty charges. The time they will last depends largely on the quantity and quality of the ash in the fuel used, and the care exercised by the men when poking the fire. At the Germania Works, with an English coke of 4 per cent. ash and careful handling, they have been made to last twenty to twenty-five; with very careful watching they have been made to last forty-five. As soon as the zinc commences to collect, a wagon, containing the moulds for the zinc and the support for the condensers, is rolled up against the front of the furnace, which has been entirely free since the charge was introduced. The zinc distils, and is collected in the condenser, and held there by the oxides and blue powder which collect in front, and are used by the workmen to form a dam to hold the zinc back. When sufficient has collected it is drawn into the moulds. The total amount collected as metal varies from 45 per cent. to 60 per cent., and is used over again. The blue powder and oxides amount to from 20 to 30 per cent.; these are sold to the zinc works. Some of the zinc is lost by volatilisation, and from 0.7 to 1 per cent. retained in the lead. The blue powder of the condensers, and the dross which forms in the retort, are difficult materials to treat. They contain some silver. As the blue powder is mostly metallic zinc, it can be mixed, to some extent, with zinc used in the first additions of zinc used on the charge or added to the blast furnace charge. It was formerly sold to the zinc furnaces, but this is not now usually done. The dross, slags, old retorts, cupelle bottoms, litharge, and most of the products containing silver are worked into the shaft furnace charges, or are sometimes treated in a special reverberatory furnace.

As soon as the zinc escaping appears in small quantity, the lead contains but little zinc; but as it is desirable to remove, as far as possible, the last traces of it, the heat is kept

up, the condenser is removed, and small pieces of wood are put into the retort to assist the discharge of the fumes. When no more escape, the furnace is tipped down and the contents of the retort discharged into a lined receiver, and there left until cool enough to be cast into pigs. They generally contain from 2000 to 3000 ounces of silver, and not more than from 0.5 to 0.8 per cent. of zinc. The retort is now carefully scraped with an iron scraper, to remove any slag or other material adhering to the sides. The amount removed in this way is not large; but it is necessary to keep the retort clean, for if the material was allowed to accumulate, it might be difficult to remove it, and there would be a risk of breaking the retorts in doing so. The material so collected, amounting usually to a few pounds, is reduced with the cupellation litharges. The unburned charcoal is put back into the retort. When the retort is cleaned, it is turned up partially, and fine charcoal dust, or a piece of wood, thrown in, to make a reducing atmosphere, and prevent the formation of litharge from the oxidation of the very small quanity of lead attached to the sides of the retort. This precaution is very necessary, for if the litharge was allowed to form, it would soon destroy the retort. The furnace is now turned up and is ready for a fresh charge. The time required for a charge of 350 lb. is from 8 to 12 hours.

The workmen are obliged to be careful in all these furnaces, that in introducing the coke they do not push too hard on the retort, which is quite soft. The fire must be kept at a constant temperature of white heat throughout the operation, which lasts from eight to ten hours according to the percentage of zinc in the alloy. But when the lead contains antimony, it lasts a much longer time.

The only precaution required during the operation is to keep the temperature high enough to prevent the formation of a crust on the surface of the charge. To prevent this, and to know what is going on in the interior of the retort, without removing the condenser, it is probed from time to time to break the crust, for if it should form, an explosion would be likely to take place. The men can always tell the condition of the heat by looking into the coke-charging hole.

It is very necessary that the current of gas should always be

out of the retort. The retort should last twenty operations on an average, and it is generally broken before it is worn out; but when much antimony is present in the lead, they last a much shorter time, so that it is always desirable to soften the metal before treating it with zinc. In some of the works, owing to careless management in not carefully cleaning the inside and outside of the retorts, they lasted for only nine to ten operations. When a new retort is necessary, the furnace must be allowed to cool down, the grate-bars are taken out, and the retort introduced from the bottom. It would seem that there should be no question with regard to the adoption of the tilting furnace. The most expensive articles used in it, however, are the retorts. In a fixed furnace like that at Cheltenham, they may, with a little care, be made to last twenty-five or even thirty charges. In the tilting furnace, however, they do not last much over fifteen. The cause of their decay is, on the inside, from the formation of lead oxide, and on the outside of clinker, which is not so easily removed in the tilting as in the stationary furnace. In some works in Europe retorts, weighing over a ton, lined on the inside with graphite, have been used in fixed and in regenerative furnaces with success. In the United States they have not, to my knowledge, been used. In some works the outside of the retort has been washed with clay, which does not allow the cinder to adhere to the sides of the retort, or makes it separate easily. In others it is said that they have been coated with a mixture of fine quartz and feldspar which effects the same purpose.

The flues leading to the chimney, at Mansfield, are made with flaring sides at the bottom, for 18 in. in height. The sides of the upper part are vertical and are rounded at the top. Every 7 ft., at the bottom, a partition is put in, one-third of the whole height of the flue. In the brick flues, which are 5 ft. high, the partitions are put in every 18 in., and further apart. In both the iron and brick flues the most dust is caught near the furnace. The dust settles by gravity in these catches, and as there can be no velocity there, owing to the partitions, it remains there. Short flues of this construction have been found to be much more effective than large condensing chambers. The experience of some of the best works in Europe is that surface, not volume,

is the indispensable requisite to the proper collection of both the particles carried off mechanically as well as those which are volatilised.*

The amount of zinc in the skimmings is very variable. If it contains 35 per cent. of zinc, 20 per cent. will be recovered as metallic zinc, and 10 per cent. as oxide, which is afterwards reduced, and 5 per cent. will be lost. This last is either in the lead or volatilised in the different operations. If the skimmings contain only 10 per cent., 3 per cent. will be recovered as metallic zinc, 5 per cent. will be recovered as oxide, and 2 per cent. will be lost. No lead or silver is found in the distilled zinc.

The tilting furnace is a very great improvement on all those in which the retort is fixed so far as the work is concerned, as it necessitates the least amount of it being done on it, and at the same time allows perfect manipulation of the furnace, but it does not always allow of using the fewest retorts.

The following Table, prepared by Mr. E. F. Eurich, gives the account of two charges in Faber du Faur's furnace, at Mansfield:†

DISTILLATION OF THE ZINC RICH IN SILVER.

—	No. 1.	No. 2.
Weight of alloy per charge with ¾ lb. of fine charcoal	353 lb.	353 lb.
No. of charges	27	20
No. of distillations in 24 hours in each retort	2	2
Total amount of liquated zinc-crusts charged	9525 lb.	6362 lb.
Charcoal	108 ,,	80 ,,
Result: Rich lead	7609 ,,	5221 ,,
Metallic scraps	390 ,,	not weighed.
Charcoal with little metal	not weighed.	,,
Metallic zinc	770 lb.	,,
Blue powder and oxide	not weighed.	,,
Coke used, in bushels of 40 lb.	410.4	276
Quantity of coke per lb. of zinc-crust	1.7	1.73

M. Faber du Faur has proposed another furnace, shown in Fig. 25, constructed on the tilting principle, which is designed

* "Collection of Flue Dust at Ems." Trans. Am. Inst. Min. Engineers, vol. ii., p. 379.

† Mining Commissioners' Report for 1875.

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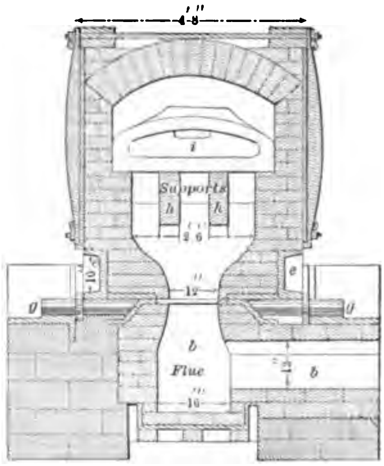


SECTION THROUGH C.D.

ndenser on truck.
derground flue.
s-escape.
s-escape-flue with a movable cover for
ning condenser.
n beams on which the furnace is built
d at f.
nnions on which the furnace moves.
e-brick supports for the retort i.
rt.

to receive a charge of 1 ton at a time. The retort, *i*, is made of fireclay, lined on the inside with graphite. It is 6 ft. 6 in. long on the outside, 5 ft. 10 in. long on the inside, and 7 in. high. It is placed on a cast-iron frame, *e*, protected by firebrick, and connects with a condenser, *a*, 12 in. in diameter and 2 ft. 3 in. high on the inside, which is placed on wheels so as to be moved when the retort is to be tilted. The retort is moved mechanically from the fireplace end at *f*. The furnace may be constructed for solid fuel, as in the drawing, but it was invented exclusively for the use of gas and hot air. The object in the construction of the retort was, to have the largest possible surface for distillation, with the shallowest depth of metal, which will not exceed 2½ in. to 3 in. It was proposed to make the retort in two parts if necessary. This furnace has never yet been built, on account of the commercial depression. Contracts for its construction were once prepared but not completed. It seems to have the advantage of being able to treat a large quantity expeditiously, and thus economise in labour and material.

The silver-lead, containing from 2000 to 3000 ounces of silver, is cupelled in an English cupelle furnace. At the Germania Works, there are two of these furnaces; at Cheltenham only one. They are blown with a steam jet in both places. They are usually at work one week, during which time they treat 35 bars of 65 lb. each per day. The silver is then tapped, and the test changed, or the other furnace used. The silver bullion produced weighs about 9000 ounces, and is usually 990 to 995 fine, and contains four to five thousandths of gold, the proportions of both metals varying with the bullion or ore purchased. The litharges produced are reduced in a reverberatory furnace. At Cheltenham the cupellation is done on a Steitz water-back cupelle made of a hollow casting, through which water flows. This was formerly made of iron, cast in one piece, but it was found not to answer, as the front which contained the litharge channel wore out rapidly. This part was then made separate, so that it could be quickly replaced when the furnace was working. Recently copper water-backs with replaceable iron litharge fronts have been used. The lead is concentrated on the water-back up to 60 per cent. silver and is then finished on an ordinary

English hearth. A very great advantage has been found in thus dividing the operation, as up to about 60 per cent. of silver the concentration can be done by an ordinary workman. The finishing only will thus have to be done by an experienced man. It is also considered that as the responsibility is more individualised, this method of conducting the operation in two parts, greatly diminishes the temptation on the part of the men to steal the silver, and the risk on the owner of losing any of it.

At Mansfield Valley, the cupelle was formerly made of an artificial marl, made by mixing clay and pulverised limestone together, as is frequently done in Germany. This was stamped into a mould and the shape of the test cut out of it. Now, however, both here and at Aurora, Illinois, the cupelle is made of the best hydraulic Portland cement, moistened enough to ball in the hand, and stamped in an iron mould. The test is 3 ft. by 4 ft. on the inside. The iron frame which supports it is flanged on the bottom at right angles to the rim, which is 7½ in. high, while the flange is 5½ in. wide. The test is made either on an iron mould, which gives the shape to the inside, or is cut out of the material after the frame has been stamped full. At first they were always cut out, now they are generally stamped over the mould. When made, the cupelles are left to temper for four weeks, to insure a good test. They could be used after a week but it is better not to do so. The test is supported in the furnace on an iron plate, and is held up to its place by four large screws. The charge of a rich alloy is 1400 lb. The cupelle is used for a week, and cupelles from 10 tons to 12 tons up to 996 fine, and that directly from the lead. The lead is added in the cupelle till just before it is too rich, then cleaned off and the silver is refined, and is run into the brick moulds directly from the cupelle. A little copper is added, to prevent the spitting of the silver. The copper absorbs the oxygen, and prevents the spitting. When any copper is present in the lead, even when gold is present it rarely ever spits. When the silver is ready to cast into bricks, the test is loosened, and a curved bar is placed on a support made for the purpose underneath it.* The whole test is then raised by a Weston pulley, and the silver, tipped at once into the moulds.

* Trans. Am. Inst. Min. Eng., vol. x., p. 220.

for the bricks, is 994 to 996 fine. This cupellc thus allows of casting, without refining in a separate furnace. It is the invention of Mr. Eurich, formerly the manager of the Pennsylvania Lead Works, and is one of the many ingenious additions to metallurgical progress which he has made.

Taking into view the fact, that lead which contains an appreciable amount of silver and the other impurities which usually accompany the precious metals when they are found in lead, cannot be so used in most manufacturing processes, what amount of silver it will be economically worth while to separate is rather a question of the relation between the price of the lead containing these impurities and marketable lead than of the quantity of silver contained in it. The value of such material may, however, be approximately estimated by the charges usually made for refining it. For ordinary silver-lead containing no gold and of not too high a grade,the ordinary refining charges would be about $13. Ninety-five per cent. of the lead, and all but 3 ounces of the silver, would be returned. This would make the cost to the owner, within the last two or three years, a little over $20 the ton. Lead with a lower silver contents, if free from other impurities, could be treated at a much lower price, but such material is rarely, if ever, found.

The following Tables, prepared by Mr. E. F. Eurich, give the results of cupellation at Mansfield:*

* Mining Commissioners' Report for 1875.

SILVER OBTAINED.

	No. 1.		No. 2.	
	oz.	oz.	oz.	oz.
Quantity of silver in the refined work lead	...	6,305.6	...	6,165.9
Silver tapped from the cupelle, .980 fine 6,088.75 oz.	6,031.66			
Silver tapped from the cupelle, .989 fine 5,714.50 oz.	...	...	5,645.9	
Small pieces of silver from the cupelle .970 fine 150.00 oz.	146.50			
Small pieces of silver from the cupelle .970 fine 115.00 oz.	...	...	111.5	
In market lead 0.33 oz. per ton in 67,104 lb.	11.18			
In market lead 0.33 oz. per ton in 53,420 lb.	...	...	8.9	
In litharge lead 30 oz. per ton in 5,209 lb.	...	...	78.0	
Total silver obtained	6,189.34	...	5,844.3	
Silver not recovered	116.26	...	321.6	
	6,305.60	6,305.6	6,165.9	6,165.6
Percentage of silver obtained	98.1	...	93.3	

LEAD OBTAINED.

	No. 1.		No. 2.	
	lb.	lb.	lb.	lb.
Work lead used	...	87,294	...	62,895
"Schlicker" 3497 lb. at 80 per cent.	2,797			
Impure litharge from dezincation, 3500 lb. at 80 per cent. lead	...	...	2,800	
Lead in zinc crust	7,765	...	5,002	
Soft market lead	67,104	...	53,420	
Oxides and skimmings from market kettles, 1000 lb. at 95 per cent. lead	950			
Oxides and skimmings from market kettles 700 lb. at 95 per cent. lead	..	...	665	
Litharge from dezincation, 7810 lb. at 80 per cent. lead	6,248			
Lead from liquation	808			
Total lead	85,672	...	61,887	
Loss about 1.9 per cent.	1,622			
" " 1.7 "	...	...	1,008	
	87,294	87,294	62,895	62,895

The following example of an operation at the Pennsylvania Company's Works has been prepared for me by Mr. J. A. Knapp, formerly the superintendent there:

Quantity of argentiferous lead from Utah ores, charged	26 tons.
,, silver after dressing in the softening furnace	55.94 oz.
Number of skimmings	2
Intervals between skimmings	6 hours
After dressing in the zinc kettle, the lead contained silver	57.63 oz.
,, the first addition of zinc	13.34 ,,
The second addition of zinc was	300 lb.
The time between No. 1 and No. 2	5 to 6 hrs.
After the second addition the lead contained silver	0.87 oz.
The third addition of zinc was	150 lb.
After the third addition the lead contained silver	0.09 oz.
Number of skimmings in softening furnace	3
The merchant lead from the softening furnace contained silver	0.093 oz.

The following Tables, 1, 2, and 3, have been prepared for me by Mr. E. F. Eurich, as the result of the work at the Pennsylvania Lead Company's Works, in August, 1879:

1. Lead Obtained.

——	Pounds.	Pounds.
Bullion Charged	...	654,074
Produced:		
Metallic dross, from refining furnace	11,402	
43,540 lb. refining furnace skimmings, at 83 per cent. lead	35,138	
Metallic dross from the desilverizing kettle	16,290	
37,357 lb. zinc crusts, containing	32,604	
27,616 lb. softening furnace skimmings, at 83 per cent. lead	22,921	
Merchant lead	533,207	
	651,562	
Loss	2,512	
	654,074	654,074

From the above amount of bullion, there was produced 591,244 lb. of refined bullion, ready for desilverizing.

The silver contents of merchant lead vary from $\frac{5}{100}$ to $\frac{10}{100}$ ounce.

The refined bullion is desilverized with from three to four zinc additions, varying according to the richness of the bullion.

The total quantity of zinc added to effect the complete desilverizing of 591,244 lb. refined bullion, was 7860 lb.

2. Silver Obtained.

—	—	Fine Silver.
Charged:	oz.	oz.
Silver contents of 591,244 lb. refined bullion .	...	35,048.23
Produced:		
Poured from cupelle 33,318 oz., containing .	33,147.75	
Contained in 29,898 lb. litharge, at 38.00 oz.	568.06	
,, ,, 1,302 lb. test bottoms . .	369.77	
,, ,, skimmings from filling test .	61.33	
	34,146.91	
Difference contained in dross from retorts and loss	901.32	
	35,048.23	35,048.23

3. Distillation of the Zinc Crusts.

—	Pounds.	Pounds.
Charged:		
Liquated zinc crust.	...	37,357
Produced:		
Lead riches	31,142	
Dross from retorts, about	2,200	
Metallic zinc, about	2,900	
Blue powder and zinc oxide, not weighed .		
Not accounted for	1,115	
	37,357	37,357
Number of charges made	59	
The average weight of charge	629	
Total coke consumed	21,000	
Quantity of coke per lb., zinc crust. . .	0.56	

The following Tables were taken by myself from the books at Mansfield Valley, with the permission of Mr. E. F. Eurich, in June, 1880, and refer to the months of April and May of that year:

——	Ounces.	Ounces.
CUPELLATION, April 1st, 1880.		
From desilverizing kettles to retorts	...	31,865.84
Retorts to cupelle	31,790.08	
Extracted from 930 lb. retort scrap	468.81	
Retort gain	...	393.05
	32,258.89	32,258.89
CUPELLATION, April, 1880.		
Charged 27,883 lb. of lead riches	30,659.58	31,790.08
Produced silver bricks shipped	723.88	
Silver scrap	430.08	
Litharge, 26,880 lb., at 32 oz.	49.60	
Test bottoms, 400, at 248	87.10	
Cupelle skimmings, 109 lb.		
Cupelle gain	...	160.16
	31,950.24	31,950.24
CUPELLATION, May, 1880.		
From desilverizing kettles to retorts	...	43,907.69
Retorts to cupelle	43,208.76	
Extracted from retort scrap	839.33	
Retort gain	...	140.40
	44,048.09	44,048.09
CUPELLATION, May 8th, 1880.		
40,130 lb. lead riches, containing	...	43,208.76
Silver scraps	...	1,315.18
Fineness samples	...	34.80
Produced silver bricks shipped	43,013.47	
Silver scrap	354.35	
Contained in 398 cupelle skimmings	261.28	
Litharge, 39, 148, at 38	743.72	
Test bottom, 826 lb., at 218	90.03	
Cupelle loss	95.89	
	44,558.74	44,558.74

The following Table, taken by myself, from the books of the company, gives a summary of the work for April and May, 1880:

—	April.	May.
Metallic dross from refining furnace, in per cent. of gross charge	.078	
0.85 of lead and skimmings	2.88	2.47
1st net weight in desilverizing kettles 1000	96.34	97.53
1st crass from ,, ,,	7.84	7.55
2nd net weight ,, ,, 96.34.	*a* 88.50	89.98
1st average assay of kettles	144.67	172.73
2nd ,, ,, ,,	*b* 132.64	160.92
1st crasses in per cent. of 1st net metal in desilverizing kettles	*c* 8.58	8.04
Retort in per cent. of 2nd desilverizing kettles	*d* 7.54	8.32
Zinc used in per cent. gross charge	1.37	1.59
,, 1st net weight in desilverizing kettles	*e* 1.43	1.63
,, per cent. of merchant's lead	1.72	1.98
Coal used per ton of gross charge	192.16	208.00
,, ,, merchant lead	238.00	262.00
Lead in merchant lead	90.47	88.99
,, retort crasses.	6.78	7.47
,, refining skimmings.	3.28	3.48
	100.53	99.94
Apparent gain, 0.53 per cent.	Loss 1.06 per cent.	

The losses and gains are apparent only.

a Charge for the retorts calculated on this. *b* Average assay. *c* Percentage of 96.34. *d* Percentage of 88.50. *e* Percentage of 96.34.

The lead made by the Germania, Pennsylvania Company, and St. Louis Works is exceedingly fine. As it can be used for the manufacture of white lead, it commands the highest market price.

The following analyses, made by Dr. O. Wurth and Dr. Zuireck, on a sample from the works of the Pennsylvania Lead Company, show that the lead is equal if not superior to any of the brands produced abroad.

In 100 Parts.	Dr. O. Wurth.	Dr. Zuireck, Berlin.	Dr. O. Wurth.	Dr. Zuireck, Berlin.
Silver	0.00042	0.00035	0.00016	0.00070
Antimony	0.00051	0.00254	0.00318	0.00346
Copper	0.00007	0.00094	0.00005	0.00093
Zinc	0.00038	0.00070	0.00122	0.00075
Iron	trace	0.00082	0.00013	0.00082
Sulphur	0.00018	...	0.00023	...
Arsenic	none	...	trace	...
Bismuth	...	0.03843	0.02746	0.04594

CHAPTER III.

SEPARATION OF GOLD AND SILVER FROM COPPER.

THE BOSTON AND COLORADO SMELTING WORKS.

THE Boston and Colorado Smelting Works were formerly situated in the town of Black Hawk, Gilpin Co., Colorado, on the Clear Creek Narrow Gauge Railway, 55 miles from Denver, in the Rocky Mountains, at an altitude of 7800 ft. It was one of the first works erected for the metallurgical treatment of gold ores and the only one established in Colorado on a large scale which has been uniformly successful. The works were planned and built by Professor Hill, formerly professor of chemistry in Brown University, Providence, R.I., and is still managed by him, assisted by Mr. Richard Pearce, formerly professor in the School of Mines at Truro, England. The works are very advantageously situated with regard to the ore-producing regions, having Boulder Co. on the north, which produces, besides ordinary gold ores, a series of tellurium minerals, such as altaite, sylvanite, and hessite, which are very rich in gold and silver. They are associated with copper and iron pyrites, blende, galena, and the oxides and carbonate of iron.

Gilpin Co. itself produces for the most part pyrites, both of iron and copper rich in gold, with a small quantity of galena and blende which is rich in silver. In some mines native gold is found. Clear Creek Co. to the south furnishes mostly galena and blende very rich in silver. The works also receive mattes rich in gold and silver made at Alma in Park Co., and tellurium ores rich in gold from the southern part of the territory.

The works are thus located in the very centre of the gold and silver-producing regions at Colorado, and are also most favourably situated with regard to transportations. They treated in 1874 30 tons of ore and tailings in twenty-four hours, and produced

700,000 oz. silver, 12,000 oz. to 15,000 oz. gold, and 225 tons copper. With matte from Alma their production in 1875 was 110,000 oz. of silver, 25,000 oz. of gold, and 250 tons of copper. In the year 1880 they produced $568,000 of gold, $2,216,000 of silver, $297,000 of copper, or a total production of $3,081,000.

Both gold and silver ores are treated, and gold, silver, and copper produced. The lead is not separated from the ores nor paid for if it exists, and is entirely lost in the residues. In the year 1878, the works removed to Argo, near Denver, where they are even more favourably situated than previously.

Gold Ores.

The gold ores are divided into three classes. The first class consists of auriferous copper pyrites containing from 2 to 10 per cent. copper, 2 oz. to 10 oz. of gold, and 2 oz. to 10 oz. of silver. These ores average 4 per cent. copper, 3½ oz. gold, and 6 oz. silver. The second class are tailings from the gold mills, consisting of pyrites with about 1½ per cent. copper, 1¼ oz. gold, and 4 oz. silver. The third class consists of tellurium ores which have a very silicious gangue, and contain 100 oz. to 200 oz. of gold, and 6 oz. to 10 oz. of silver. These ores come mostly from Boulder Co., and are often worth $10,000 to $15,000 to the ton.

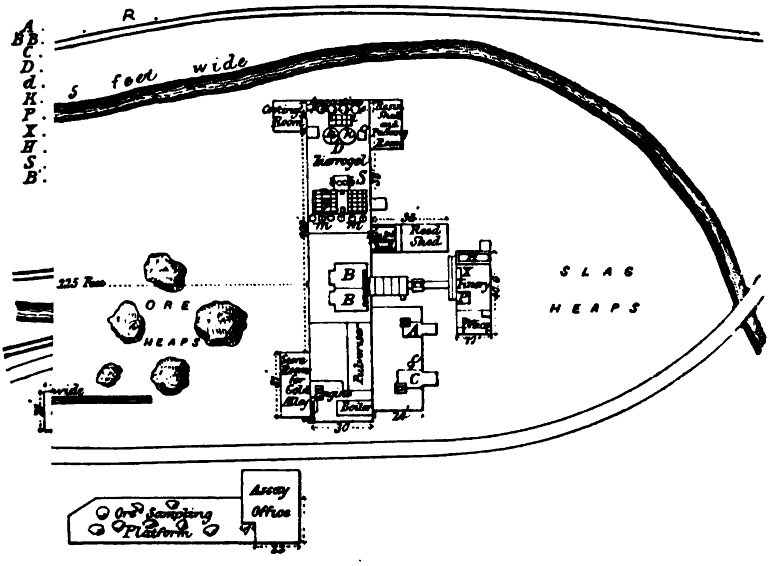
Silver Ores.

The silver ores of the first class consist of surface ores, mostly free from sulphur, containing 70 per cent. silica. They contain 100 oz. silver and 5 to 6 per cent. lead, and no gold. The second class are sulphurets rich in blende and poor in galena and pyrites; they contain 150 oz. of silver, 15 per cent. of zinc and lead, and no gold.

The cost of material at the works is :

	$
Wood per cord	5.00
Firebrick per thousand	90.00
Common brick	14.00
Iron castings per pound	.08
Wrought iron	.08

The cost of delivering the silver in New York is 1¼ per cent. of its assay value, taken at the valuation of the works. The rate



A
B B
C
D
d
K
P
X
H
S
B
R
5 feet wide
S L A G
H E A P S
O R E
H E A P S
B
B
A
C
D
S
X
Reed Shed
Boiler
Engine
Assay Office
Ore Sampling Platform
30

To

for gold is $\frac{6}{10}$ per cent. The general plan of these works is given in Fig. 26. The diagram on the next page indicates the various processes, showing what becomes of each of the products in the different stages. The treatment is composed of eight distinct operations, most of which are more or less subdivided. These operations are:

1. Sampling the ore.
2. Roasting the ore.
   - A. Large ore roasted in heaps.
   - B. Small ore roasted in a reverberatory furnace.
3. Fusion for matte.
4. Ziervogel's process.
   - A. Crushing and roasting the matte for sulphate of silver.
   - B. Precipitation of the silver.
   - C. Washing and fusing the cement silver.
   - D. Precipitating the copper.
   - E. Refining cement copper.
5. Treatment of the Ziervogel tub residues.
   - A. Fusion for white metal.
   - B. Roasting white metal.
   - C. Treatment of the pimple metal.
6. Treatment of the residues of the Ziervogel process by the Augustine process.
7. Treatment of the bottoms.
8. Treatment of the copper alloy.

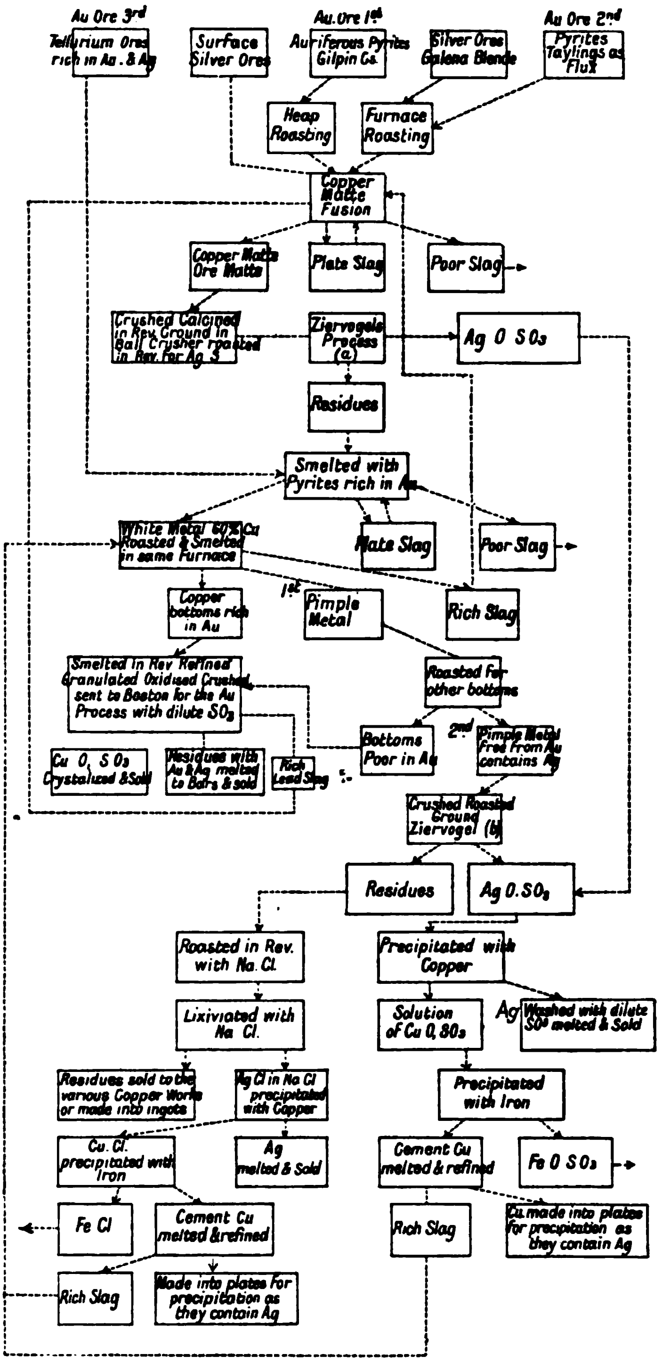
I. Sampling the Ore.

The ore is purchased in large and small sample lots varying from 50 lb. to 6 tons or 7 tons. It is sampled by first taking one-tenth of the lot, and putting it through a Dodge crusher and a pair of Cornish rolls, and then sampled as usual.

The following prices are paid by the Boston and Colorado Smelting Company, for ore delivered in car-loads at their works at Argo, Colorado, in the year 1885.

Gold and Silver Ores carrying no Copper, less than 10 per cent. of Lead or Zinc, and less than 5 per cent. of Arsenic.

Ore Assaying	Percentage of Assay Value of Gold and Silver.	Less per Ton. $
Less than $100 per ton of 2000 lb.	90	15
From $100 to $200 ,,	92	15
,, $200 to $300 ,,	94	15
Over $300 ,,	95	15
Ores carrying copper	90	15


Au Ore 3rd
Tellurium Ores rich in Au. & Ag
Surface Silver Ores
Au. Ore 1st
Auriferous Pyrites Gilpin Co.
Silver Ores Galena Blende
Au Ore 2nd
Pyrites Taylings as Flux
Heap Roasting
Furnace Roasting
Copper Matte Fusion
Copper Matte Ore Matte
Plate Slag
Poor Slag
Crushed Calcined in Rev. Ground in Ball Crusher roasted in Rev. for Ag S
Ziervogels Process (a)
Ag O SO3
Residues
Smelted with Pyrites rich in Au
White Metal 60% Cu Roasted & Smelted in same Furnace
Plate Slag
Poor Slag
1st
Copper bottoms rich in Au
Pimple Metal
Rich Slag
Smelted in Rev Refined Granulated Oxidised Crushed sent to Boston for the Au Process with dilute SO3
Roasted for other bottoms
2nd
Bottoms Poor in Au
Pimple Metal free from Au contains Ag
Cu O, S O3 Crystalized & Sold
Residues with Au & Ag melted to Bars & sold
Rich Lead Slag
Crushed Roasted Ground Ziervogel (b)
Residues
Ag O.SO3
Roasted in Rev. with Na. Cl.
Precipitated with Copper
Lixiviated with Na Cl.
Solution of Cu O, SO3
Ag
Washed with dilute SO3 melted & Sold
Residues sold to the various Copper Works or made into ingots
Ag Cl in Na Cl precipitated with Copper
Precipitated with Iron
Cu. Cl. precipitated with Iron
Ag melted & Sold
Cement Cu melted & refined
Fe O S O3
Fe Cl
Cement Cu melted & refined
Rich Slag
Cu made into plates for precipitation as they contain Ag
Rich Slag
Made into plates for precipitation as they contain Ag

Ores containing less than 10 per cent. of Lead or Zinc and less than 5 per cent. of Arsenic containing Gold and Silver.

Over $8.00 a ton in gold and silver for one per cent. of copper	$ 1.00	per unit of copper.
From $5.00 to $8.00...	.90	,, ,,
Below $5.00	.80	,, ,,

No charge is made for crushing, sampling, and assaying the ore if in car-loads ; if there is less than a car-load a small charge is made for handling. They purchase no ores with over 10 per cent. of lead. Ore assaying less than this is purchased according to the scale given above, no allowance being made for the lead. Special prices are given for gold or silver ore carrying iron or copper pyrites, and less than 10 per cent. gangue and also for matte and other furnace products. Deductions are made from the scale of ore having a large percentage of zinc for arsenic.

When the ores are sent by the owner for treatment the charge formerly was:

	$		$
For ores containing	50	...	35
,, ,,	60	...	36
,, ,,	70	...	37
,, ,,	80	...	38
,, ,,	100	...	40
,, ,,	200	...	50
,, ,,	300	...	60

and so on.

The arrangement of July, 1874, was that the company should pay for gold ores at the rate of 85 per cent. of the total value of the gold and silver contained, the premium being added after deducting $35 per ton currency for treatment. The gold was estimated at $20 gold per ounce, and the silver at $1.25 cents gold per ounce, with the premium added 3 per cent. below New York quotations.

The arrangement for silver at the same time is given in the Table annexed. The prices are based on the premium on gold in New York ranging between $1 10 cents and $1 15 cents.

—	Ounces per Ton.	Cents per Oz. in Currency.	—	Ounces per Ton.	Cents per Oz. in Currency.
For ores containing	40	34	For ores containing	300	101
,, ,,	50	44	,, ,,	350	103
,, ,,	60	52	,, ,,	400	105
,, ,,	70	60	,, ,,	450	106
,, ,,	80	66	,, ,,	500	107
,, ,,	90	70	,, ,,	600	108
,, ,,	100	74	,, ,,	700	109
,, ,,	125	82	,, ,,	800	110
,, ,,	150	89	,, ,,	900	111
,, ,,	175	93	,, ,,	1000	112
,, ,,	200	97	,, ,,	2000	116
,, ,,	250	99			

The copper is paid for at $1 50 cents currency for each unit by the dry Cornish assay.

The precious metals in the ores were formerly never paid for above a certain minimum, which for silver was 40 oz. and for gold 1½ oz. All above this minimum was paid for in currency at the rate of 85 per cent. of its bullion value. For all copper above two per cent. $1 50 cents currency was given for each unit. All the ores received are piled separately on the sampling ground.

All the large pieces of gold ore are roasted in heaps, and are then passed through a crusher and rolls, and afterwards through a screen with four-to-the inch mesh. The tellurium ores are only crushed and passed through a ten-to-the-inch mesh screen, and are then ready for smelting. The surface silver ores are crushed, and passed through a four-to-the-inch mesh screen, and then go to the furnace. The ores rich in sulphur are called heavy ores and are crushed and calcined in a large reverberatory furnace.

II. Roasting the Ores.

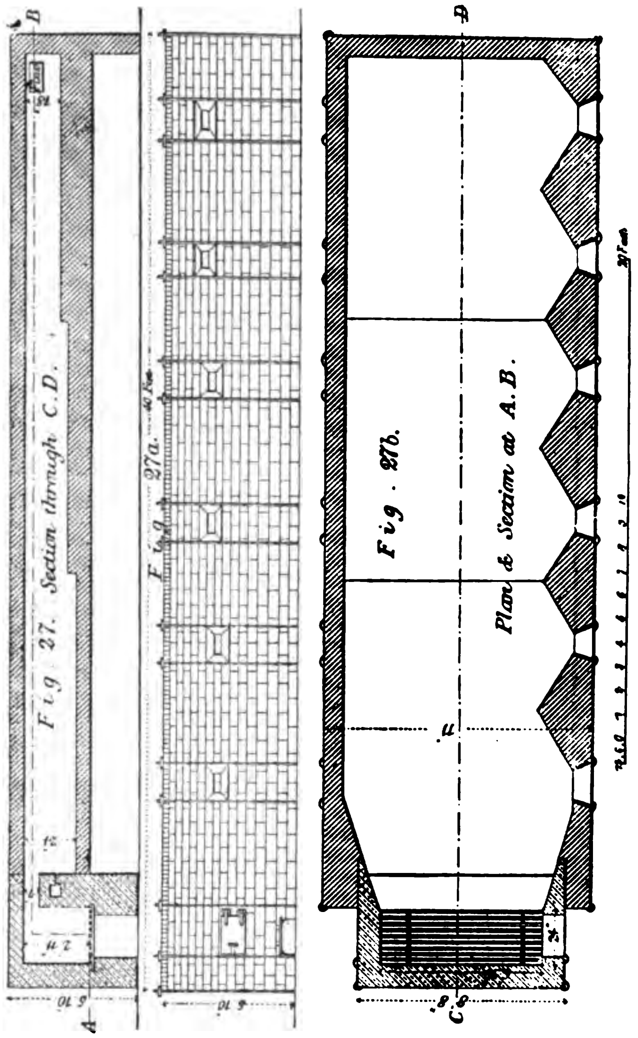
A. *Roasting Ores in Heaps.*—The auriferous pyrites are broken to 2 in. square in a crusher and roasted in heaps of about 50 tons each. The piles are made in the usual way with a wooden chimney about 7 ft. high in the centre. Wood is used

as fuel. The amount consumed is two cords for 50 tons. The wood is burned out in about twelve hours, at which time the sulphur commences to burn. The pile is lighted at night, because the moisture in the fuel makes sulphuretted hydrogen, which would annoy the men in the daytime. The fire, except in case of accident, burns until the roasting is complete. The sampler takes charge of the piles. He has little to do except to throw fine ore on the cover when he sees that there is too much flame. He has two or three assistants, and with them he does all the weighing and sampling, and takes care of the piles. When the pile is finished the outside crust of unburned pyrites is taken off and put on to the next pile. The roasted ore is crushed and goes through a sieve with four-to-the-inch mesh, and is then ready for the smelter. One man does the whole crushing. The roasting is finished in about six weeks from the time the fire is lit. The amount of sulphur remaining in the ore is 4 per cent. As the ore contains considerable arsenic the pile is frequently covered on the outside with crystals of arsenious acid, which are often white but generally coloured with a slight trace of sulphur or sulphide of arsenic. They are generally found when there has been a hole in the cover of the pile, and their usual form is that of an octahedron with hollow faces.

B. *Roasting the Ore in a Reverberatory Furnace.*—The ore submitted to this process is said to be calcined.* The tailings and finely divided copper ores are roasted in a reverberatory furnace called a calciner till they contain not more than one-half to 4 per cent of sulphur. There are six of these calciners in the works. They are marked K on the ground plan, Fig. 26, and are shown in detail at Figs. 27, 27*a*, and 27*b*. Only three of them are in use at a time, two of these work into the same flue. Each furnace has three step-hearths 10 ft. long. The total length of the furnace is 40 ft. on the outside, including the fireplace. They are 11 ft. wide, and have six working doors, two doors to each hearth.

* The word "calcining" as used in these works means treating the ores or mattes in a fine state in an oxidizing atmosphere. The term "roasting" means the treatment of metal ingots or mattes in large pieces in an oxidizing atmosphere. As these words are not a proper use of metallurgical terms we shall speak of both as roasting.

The hearths are 4½ in., the one above the other, and are equally divided in the length of the furnace. Each one is rectangular,



REVERBERATORY FURNACE FOR ROASTING ORES.

with the usual waste space at the doors, filled up. The two at the end have their corners rounded. On comparing the relative dimensions of these furnaces, it will be seen that the surface of the hearths is 304 square feet. The surface of the grate is 16 square feet. If the fireplace is taken as unity, the relation between the surfaces will be as 1 : 19. The fireplace is

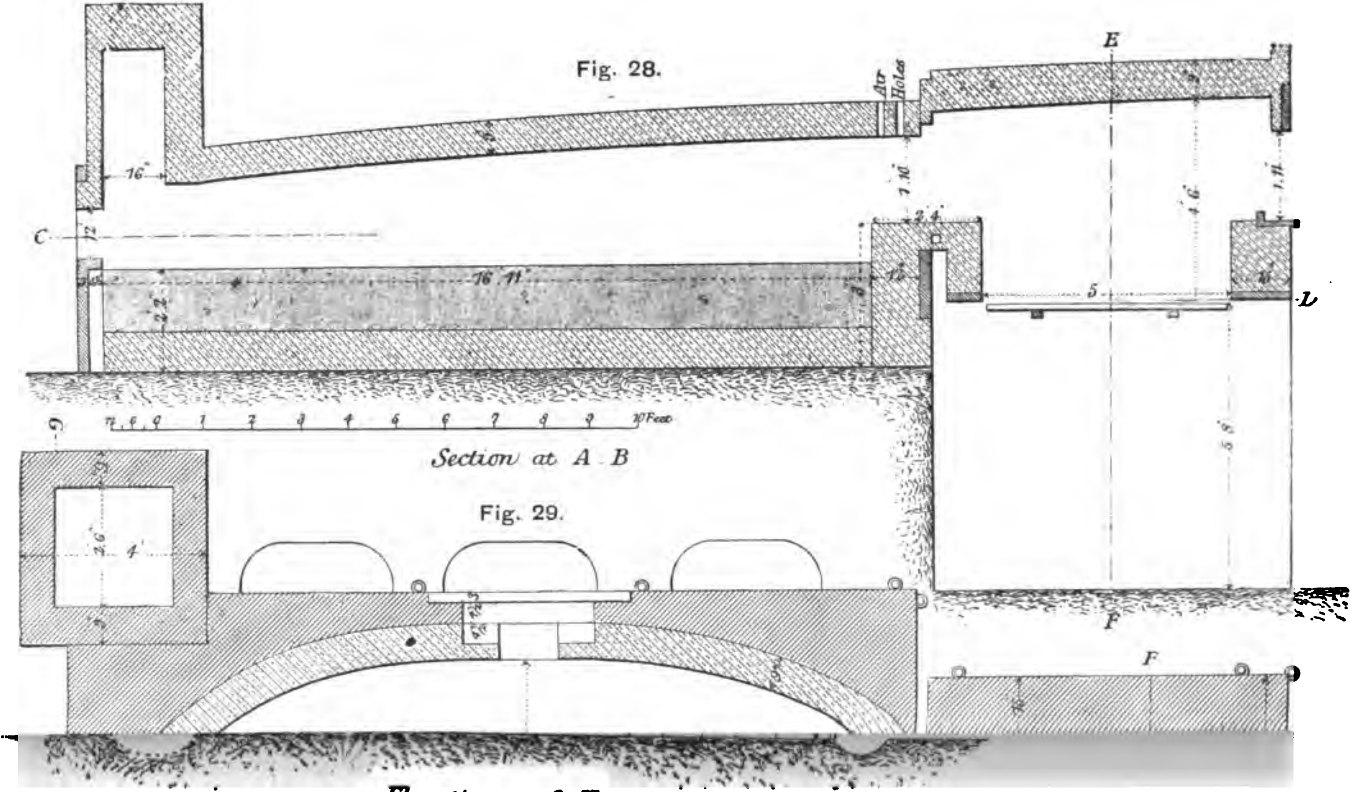


Fig. 28.

Section at A B

Fig. 29.

Elevation at G. H.

To face page 128

arranged for long sticks of wood, and has a door at the side. It is 5 ft. long and 2 ft. 8 in. wide. The bridge has an air-hole in it which is 4½ in. square, and communicates with the interior of the hearth by four openings. The width of the bridge is 28 in., the height of the roof above the hearth is 28 in., and at the flue end it is 18 in.

The furnace is built of red brick, firebrick being used only in the fireplace and on the first hearth. A charge of one ton is introduced on the hearth nearest the flue, so that there are three tons in the furnace at a time. The charge of ore on each hearth, spread out after being first put in, is 3 in. deep, but it swells, so as to be 4 in. to 5 in. deep, on the hearth nearest the fireplace; this is particularly true of the tailings. As the charge is drawn once in eight hours, it takes twenty-four hours to complete the roasting of one ton of ore. One man to a shift who brings his own wood, is all the labour that is required, so that two men work three tons in twenty-four hours. The ore is brought to the furnace men, who then make the charge. One man brings all the ore for three furnaces. The men from the calciners always assist in charging the calcined ores into the matte furnaces. The furnace burns 1½ cords in twenty-four hours. One day in a year is all the repairs that are needed to the furnace.

III. Fusion for Matte.

The roasted ore is fused in a reverberatory furnace for matte. There are three of these furnaces which are marked D in the ground plan, and are given in detail in Figs. 28, 29, 30, and 31. Only two of them are in use at a time. They are constructed to use wood, so that the fireplace, which is 5 ft. at the top of the bridge, is only 2 ft. 6 in. at the grate; it is 5 ft. long, and 4 ft. 6 in. deep from the grate to the roof. The opening in the fireplace for charging fuel is at the end of the furnace, and not at the side as it usually is. The fireplace door is of cast iron; it slides in a groove, and is counterpoised with a weight. The bridge is 2 ft. 6 in. wide, the fireplace side is 2 ft. 3 in., and the laboratory side 1 ft. 10 in. from the roof. Just above the bridge there are a series of openings, 3 in. by 1 in., for the admission of air in the roof, which follow on the roof the contour of the laboratory in two rows, the

outside having eight and the interior eleven holes each. The laboratory is 15 ft. 7½ in. long by 9 ft. 9 in. wide. The working door is at the end; the two openings at the side are closed for this operation. In comparing the relative dimensions of the furnace we find that the surface of the fireplace at the height of the bridge is 25 square feet, that at the grate is 12½ square feet. The laboratory has 143.18 square feet, so that the fireplace being taken as one, the relation is as 1 : 5.7.* Each one of these furnaces has its own chimney, which is 50 ft. high. The arrangement of the holes in the roof is a very ingenious one, for as the fireplace is very deep, and is constantly filled with long sticks of wood to a depth of over 3 ft., the wood distils and forms gas, which is burned by the air entering through these holes. Before this method was introduced by Professor Pearce there was not sufficient air to produce a perfect combustion. The immediate effect of its introduction was the saving of fuel, and more equal distribution of heat. Formerly the flue connected with the chimney was constantly burning out, and needed frequent repairs.

An opening has recently been made in the foot of the chimney for the introduction of cold air, both because the combustion is better regulated, and because the cold air is mixed with the products of combustion on leaving the furnace; the repairs to the furnace are very much diminished. The hearth of the furnace is slightly inclined towards the working door, and also to one side. It is made of two layers of brick, upon which fine quartz sand is placed, which is mixed with a small quantity of wood ashes, and then agglomerated.

When the hearth is made the temperature is lowered, and the charge is introduced. The charge is made up of crushed heap roasted gold ores.

	lb.
Roasted tailings	2000
Oxidized silver ores	1500
Roasted silver ores...	1500
Raw pyrites...	300
Fluor spar...	350
Rich scorias	500

* In all of these fireplaces with inclined sides the surface taken as unity is the section at the bridge. As the grate surface is smaller the relation between the grate and fireplace surfaces should also be considered.

After the charge is drawn the furnace is repaired, if necessary, with clay, which is beaten in with a ladle-shaped instrument attached to a long handle. Such repairs are usually not made oftener than twice a week. The charge is introduced with a shovel by a side door. The ore is introduced first and then the rich slags. The charge is so arranged that ten tons of mixed ores will produce one ton of matte. It will not do to make the matte richer, as there are always grains of it in the slag, and the loss would be greater. The slag is carefully calculated, so that it shall not be too basic, or otherwise it would cut the firebrick to get silica. The charge is evenly distributed over the surfaces of the hearth, which is almost at a cherry red heat. It takes six men, working in groups of three at a time, nearly a quarter of an hour to make the charge. As soon as it is made, the charging door is built up and luted or closed with sand. The fireplace is then charged, and the furnace is left with the full power of the draught for five or six hours. During this time the workmen clean up the slag bed and tend to the fire, which requires looking after every twenty minutes. At the end of this time they stir the furnace carefully five or six minutes to bring up everything from the bottom, which should be perfectly smooth to the tool passing over it. This produces the reactions. The furnace is now left in repose for twenty minutes to effect the separation of the scoria and the matte. If lumps are found the stirring is done again, and kept up during the firing, or for about an hour. The slag is now drawn with a rabble into moulds prepared for it. The operation of skimming the slag takes about twenty minutes. When the door is open to skim the slag the latter is quite hot and fluid, and there is a constant but quiet ebullition of sulphurous and sulphuric acid, the bubbles being about 1 in. in diameter, and quite uniformly distributed. Professor Pearce asserts that the larger part of the gas is sulphuric acid. At the close of the skimming, as the slag becomes cooler, the bubbles become larger and less uniform. Just before the skimming, pieces of sheet iron, 3 ft. by 2 ft., are placed in front of the slag bed and to one side of it, to protect the workmen from the heat. The casting bed is made 10 in. deep in front of the furnace to receive the plate slag, which ordinarily contains all the grains of matte.

This casting bed has fourteen divisions, which are connected one with the other. When the slag, which covers the matte to the depth of about 3 in., is being skimmed, it is very easy to distinguish the matte below, which shows of a dark colour and a more or less brilliant surface. As the rabble goes backward and forward, the slag does not close at once over it, and the surface is exposed for a very short time. When all the slag is drawn off, a new charge of ore is introduced. Four charges are made in twenty-four hours. During each one of the operations the stirring and rabbling are conducted in exactly the same way. While the slag is tapped the matte is left to accumulate, and is tapped only once in twenty-four hours. When the matte is to be tapped, all the doors of the furnace are opened so as to chill the last part of the slag a little, so that it will not flow out from the tap-hole. It is then tapped and made into plates 3 ft. long, 14 in. wide, and 4 in. thick in the middle, the bottom being rounded. No slag flows out with it because it is too much chilled. When all the matte has been tapped, the tap-hole is closed with damp sand. The charge makes about fourteen plates. The operation of tapping the matte and stirring takes half an hour. Three men per shift of twelve hours are required to work two furnaces. Eight cords of wood are consumed in twenty-four hours. The plate slag contains on an average 5 per cent. of copper, but is often poor enough to be thrown away with the other slags. It is generally a silicate of protoxide of iron, but is sometimes more basic. The poor slag contains about 7 oz. of silver and a trace of gold. It is too poor to treat, and is thrown away. All the slag richer than this is put back into the furnace. The matte contains from 25 to 30 per cent. of copper, 20 oz. to 30 oz. of gold, 600 oz. to 1000 oz. of silver, and some iron, lead, zinc, and antimony. When the hearth bottom of the matte furnace becomes loose and rises, as it sometimes does, the whole hearth material is taken out, crushed, and treated as ore. The flues of the furnace have to be repaired every two or three months. The roof is made over once a year. The outside walls last a number of years before it is necessary to rebuild the furnace. There are produced from this fusion the copper matte which passes to the next operation, the plate slag which is put

immediately into the furnace, and the poor slag which is thrown away.

IV. Ziervogel's Process.

A. *Crushing and Roasting Matte for Sulphate of Silver.*—The matte produced from the previous operation must be roasted, and for this purpose it must be crushed fine. It is first broken up with sledges and then crushed in a Dodge crusher, with which one man can crush about 10 tons in a day. After crushing it is put through a twelve-to-the-inch screen, and is then wheeled in a wooden barrow to the calciners marked K on the general plan, Fig. 26, where it is roasted for twenty-four hours, a charge being drawn every eight hours. The charge is one ton on each hearth, so that there are three tons in the furnace at a time. One furnace working constantly does the whole work of the establishment; 90 per cent. of the sulphur is removed in this operation. The roasted matte contains about 5 per cent. sulphur, partly as sulphides and partly as sulphates. On the hearth where the matte is charged the furnace is dark. This is necessary to prevent fusion, as there must be rapid oxidation at the lowest possible temperature. When the workman is not attending to the fire he is always rabbling the charge. When the charge on one hearth is finished it is moved to the next one by a spadelle. On the middle hearth the heat is very dull, and from this temperature it is gradually raised until it is withdrawn from the furnace; on the last hearth the temperature is a bright cherry-red heat. The charge is drawn with a rabble into a "cub" beside the furnace. As there is but a small amount of sulphurous acid given off, the roasted matte remains here until it is cool enough to be wheeled in wooden barrows, when it is taken to the ball pulveriser. One and a half cords of wood in twenty-four hours is all the fuel used in this operation.

The ball pulveriser consists of a stationary horizontal sheet-iron cylinder 4 ft. in length and 2 ft. 8 in. in diameter, inside of which another cylinder of less diameter revolves. This inside cylinder is made with a cast-iron head-piece, into which cast-iron bars are fitted so as to leave a space $\frac{1}{64}$ in. between them. These

bars are kept in position by a flange and wedges, and the heads are then securely bolted together. The material to be ground is introduced into the revolving cylinder through a trough in its axis. This cylinder or grinder contains one-half ton of iron balls, which when new are 3 in. in diameter.

The cold calcined ore from the cubs is thrown on to the crusher floor, and shovelled into bins, from which it is carried by an endless chain to a hopper which communicates with the charging trough. The charge and balls revolve together at the rate of thirty-seven revolutions per minute. The ore, which is ground sufficiently fine, passes through the spaces between the bars, and falls into the stationary cylinder, which is hopper-shaped at the bottom, and communicates with a trough, through which an endless chain passes and carries the ore to a 60-mesh screen; what remains on this screen is carried back again to the grinder. The crusher works between three and four tons in twenty-four hours, and has besides plenty of time for the necessary stoppages for repairs. Six tons might easily be put through in ten hours, but from three tons to four tons is all that is required, so that a single crusher is more than sufficient. Very little repair is done to the machinery. The bars wear, and when the openings become too wide, new bars are put in. Not more than 500 lb. of balls are worn out in the course of a year. The men who do this work are obliged to wear wet sponges over their mouths in order to protect themselves from the dust. One man, who also carries the wood to the calciners, brings the ore, and one man who shovels the ore and tends the grinder, are all that is required for the work.

Roasting for Sulphate of Silver.—From the ball grinder screens, the ground matte is deposited in a bin ready to be roasted for sulphate of silver. The furnace in which this operation is conducted is called the fine calciner. There are two of them marked B B in the general plan, Fig. 26, and shown in Figs. 32, 33, and 34.

They connect with four small dust chambers which are common to both furnaces, and connect with the same chimney. They are constantly in use except when silver is being melted, when only one of them is run. They have but one hearth, which is 11 ft. 6 in. long, 3 ft. 6 in. deep, and 10 ft. 6 in. wide. This hearth is

Fig 32.

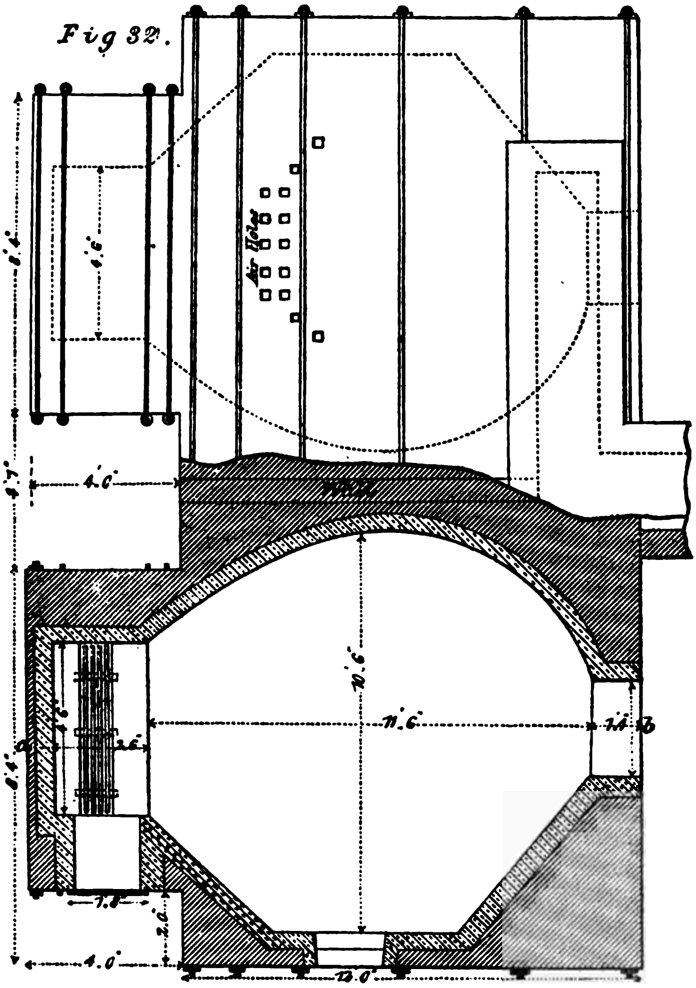
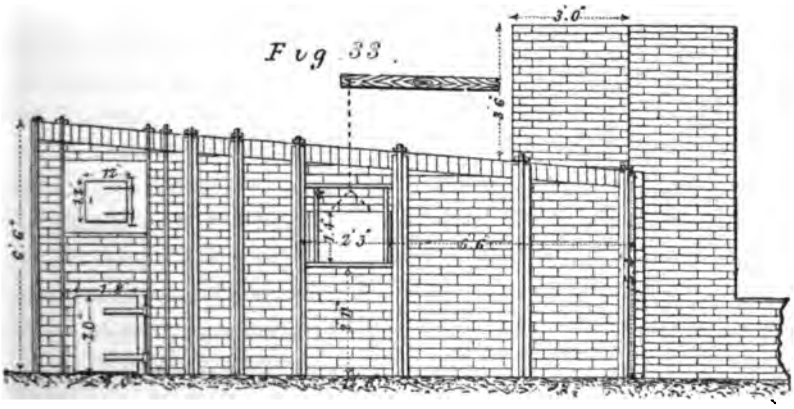


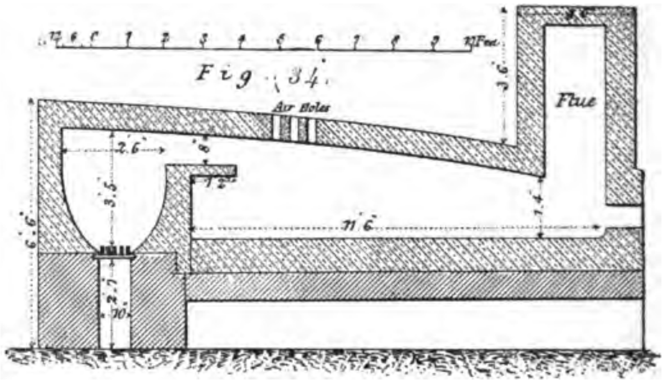
Fig 33.



FINE CALCINING FURNACE.

flat. The fireplace is 4 ft. 6 in. long, 2 ft. 6 in. wide at the bridge. The grate is only 1 ft. wide. There are 11.25 square feet surface in the fireplace, and 100 square feet in the laboratory, making the relation as 1 : 9. The top of the bridge is 8 in. from the roof. The bridge is 2 ft. wide, but 14 in. of this width is a curtain arch, the bottom of which is 16 in. above the hearth. Just

Scale for Figs. 32, 33, and 34.



FINE CALCINING FURNACE.

beyond the curtain, in the roof of the furnace there are a series of holes for the admission of air, of the same size as in the matte furnace. The first line goes straight across the roof and is composed of five holes. The second follows the contour of the furnace, and is composed of nine holes. The hearth of this furnace is made on a bed of old slag or stone covered with sand. On hese, bricks placed on end and laid in cement are placed, which form the hearth proper.

The charge is 1600 lb. of roasted crushed matte, which is thrown in with a shovel, and made into a pile on the centre of the hearth. Just before it is introduced all the dampers are closed. The hearth of the furnace at this time is dark. The fireplace is, however, glowing, but contains only embers just sufficient to keep it hot. As soon as the charge is introduced it is levelled with the rabble and spread out over the hearth. When spread out it is about 3 in. thick. It takes about twenty minutes to do this work, during which time the dampers remain closed and no fuel is put into the fireplace. As soon as the charge is completed the damper is slightly raised but no fuel is charged.

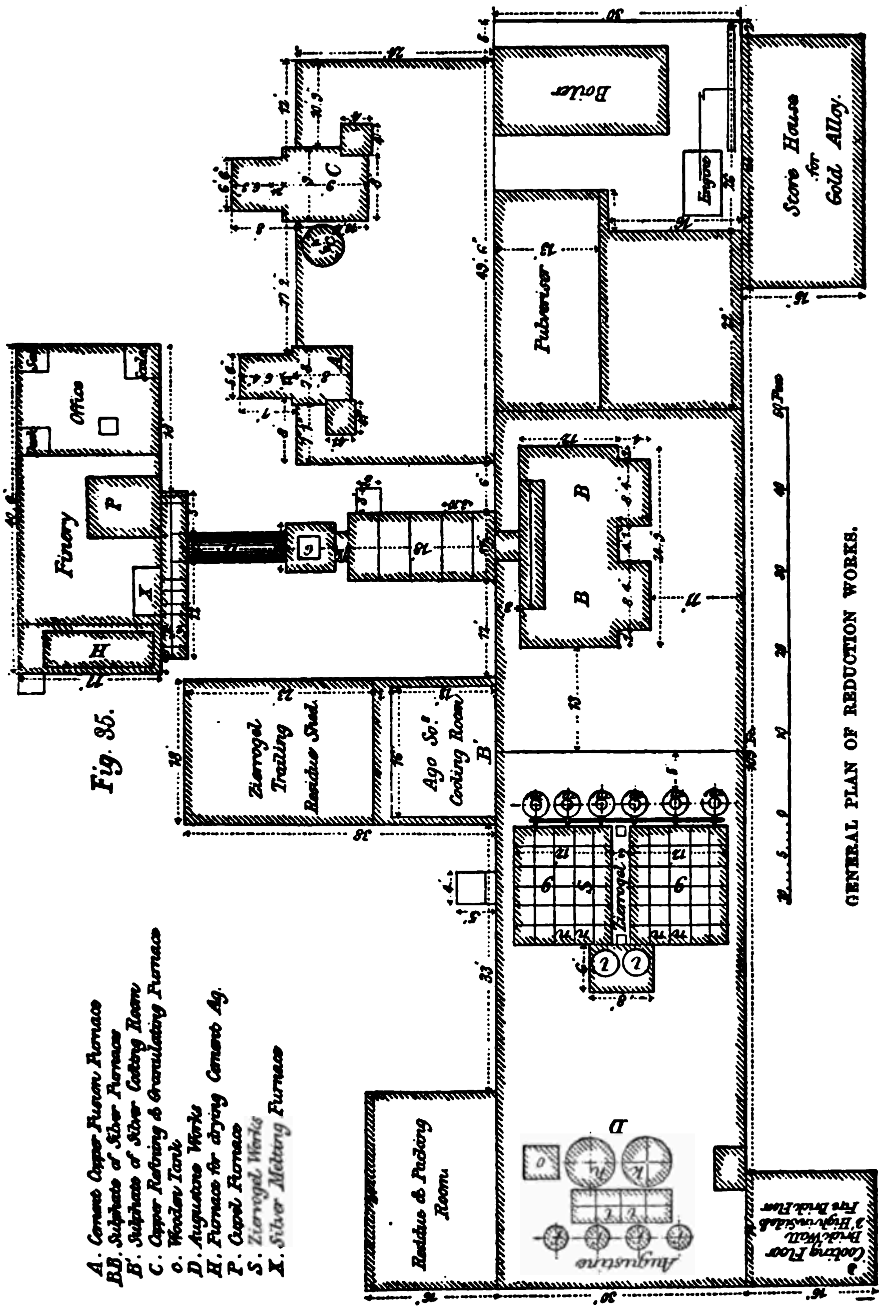
In about an hour the charge has a dull blackish glow. The surface looks black but it is red when stirred. The fireplace is now charged with a small amount of fuel, and the temperature gradually raised so as to keep it at about a dull red heat, but raising it slightly. The fireplace door is closed. The supply of air comes from the bridge holes, the working door, and the grate. The work at this stage consists of forming a maximum amount of sulphate of iron and some sulphate of copper, but the silver remains unchanged. The fumes of sulphuric acid commence to be given off from the decomposition of the persulphate of iron, and the charge increases in volume, becoming spongy. As the furnace door is open the workman is exposed to the acid fumes, and is therefore obliged to wear a respirator. The stirring is kept up and the heat gradually increased. From the second hour the grate is kept full until the end of the operation, the temperature being kept as uniform as possible. The ashpit door is closed after the first hour, the air entering only through the working door and the holes in the bridge. The flame over the curtain arch is curly, blackish, and reducing, but as there is more than 14 in. between it and the charge below, and the working door is constantly opened, it is so fully mixed with air, that in contact with the charge it is oxidizing. At the end of this time the heat is at its maximum, and the charge becomes dry, no longer sticking to the rabble.

At this point, which is at the end of three hours, the sulphate of silver is formed. The sulphate of iron is decomposed at the end of two hours. The sulphate of copper, at the time all the iron is decomposed, is at its maximum, which is at the end of the third hour. When the silver is "out" a bar 2 in. square and 14 ft. long is used to break up any lumps. The charge is collected with it into the middle hearth. The pile is then, by a sliding motion of the bar on its side, cut down, bruised, stamped, and broken up, and in this way turned over twice from one side of the furnace to the other. In order to facilitate this work, the front of the working door is provided with a roller on which the bar rests. The whole charge by this means is ground fine, and all the lumps broken up, and a perfect oxidation secured. It is essential to have as little sulphate of copper as possible, but

about $1\frac{1}{2}$ per cent. is left so as to be sure that no sulphate of silver is decomposed. This operation with the bar lasts one hour, so that at the end of four hours the charge is ready to be withdrawn. At the end of the third hour assays commence to be made, and are constantly taken until the end of the operation. The first assay generally shows that the sulphate of silver is free, but it is reduced almost instantly to a metallic state by the suboxide of copper present, and spangles are formed which scintillate and sparkle, forming a most beautiful reaction.

To make the assay, a sample of the hot charge is simply thrown into cold water in a small dish, so that its temperature is raised to above boiling. Whatever silver is in the state of sulphate is dissolved by the boiling water. If there is any suboxide of copper present, the spangle reaction takes place. At the end of the fourth hour the exposure of the surfaces to oxidation from the action of the bar has converted all the copper from suboxide into protoxide, and no spangles are seen in the assay. The sulphate of silver consequently remains permanent. If any sulphide of silver was present in the charge, it is attacked by the sulphuric acid given off by the decomposing sulphates, and converted into sulphate. An average of from 90 to 95 per cent. of silver is thus rendered soluble, the rest being in a condition of arsenides antimonides, or as very fine particles within sulphate of lead, and is not decomposed. The charges are constantly assayed, and the workmen, as they are skilled men, feel it for their interest to conduct the operation properly. It would not be safe to decompose the whole of the sulphate of copper, since there would be danger that some of the sulphate of silver would be decomposed and pass into the residues. The copper gives a blue colour to the solution, so that when the spangles are no longer produced, and the liquor is a very pale blue colour, the charge is drawn. None of this work is done at night, as the operation is exceedingly delicate, and requires to be constantly watched. As soon as the charge is withdrawn, the furnace is cooled by opening the doors and dampers to get ready for another charge. Only two charges a day are made in the furnace. It takes about ten minutes to discharge the furnace. The charge is drawn with a rabble into an iron barrow, and is wheeled to the brick cooling floor shown

Fig. 35.

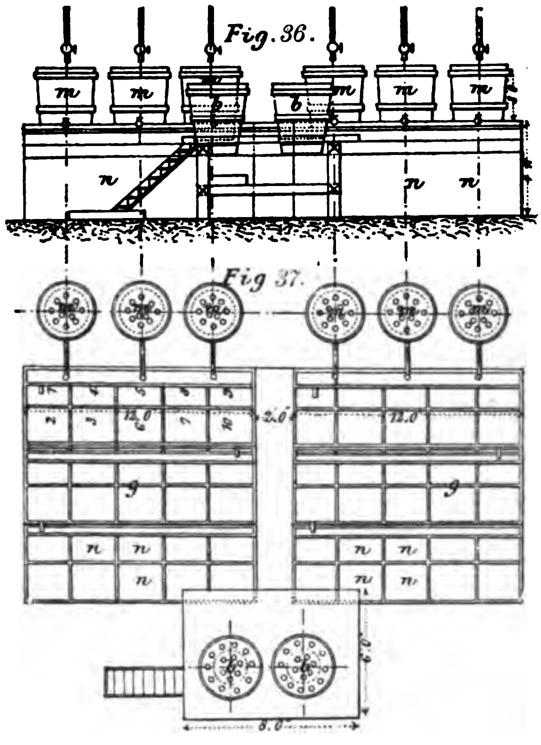


A. Cement Copper Fusion Furnace
B.B. Sulphate of Silver Furnace
B'. Sulphate of Silver Cooling Room
C. Copper Refining & Granulating Furnace
o. Wooden Tank
D. Augustine Works
H. Furnace for drying Cement Ag.
P. Cupel Furnace
S. Ziervogel Works
X. Silver Melting Furnace

GENERAL PLAN OF REDUCTION WORKS.

at B′ in the general plan, Fig. 35. Each furnace is tended by one man only. The two furnaces burn together 1¼ cords of wood in twelve hours; they require only one day's repair in a year.

Leaching the Sulphate of Silver.—The charge from the sulphate of silver furnace is allowed to remain for twelve hours on the cooling floor and is then leached in tubs. These tubs, *m*, are 3 ft. high, 3 ft. in diameter at the top, and 2 ft. 6 in. at the bottom, Figs. 36 to 38. They are provided with a double bottom pierced with holes and a cloth filter. They are charged with 1500 lb. of

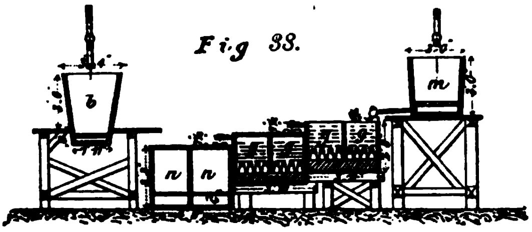


LEACHING AND PRECIPITATION VATS.

the matte which has been roasted for sulphate of silver. The leaching is done by a current of boiling water kept hot by steam. The tubs are kept constantly full and discharged into a series of tanks below. It takes eight or nine hours to leach the charge; at first it is light, but in about an hour it shrinks and the water passes less freely through it.

The residues in the tubs contain all the gold and some silver

which has not been separated. They are taken out and put on one side to be treated by the Augustine process to separate the silver, and the residues are afterwards treated for gold.* All the sulphate of copper is dissolved out in the first stages of the work. In about seven or eight hours assays of the liquid are made and the hot water stopped, when salt added to it shows no trace of silver. The time required varies according to the richness of the matte. Between 600 lb. and 700 lb. are leached in eight hours; generally about an hour is required for every hundred ounces of silver contained in the matte.



PRECIPITATION TANKS.

B. *Precipitating the Silver.*—The hot water charged with the sulphates of silver and copper from the solution tubs is run into a series of vats shown in Figs. 35, 36, and 37, and on the general plan, Fig. 26, at S.

These vats are 12 ft. long, 4 ft. wide, and 2 ft. 3 in. deep. Two of these vats, one in front of the other, are placed before each series of tubs. Each of them is divided into ten compartments, which are 24 in. by 20 in. and 27 in. deep. The liquid is discharged from the tubs into number one and communicates with two at the bottom. The partition between two and three is low at the top, so that the liquid overflows into four, and four communicates with five at the bottom, and so on. At ten the overflow passes into the tank below and follows the same circuitous course. Each compartment in the tanks is filled as shown in Fig. 38, with plates of copper ¼ in. thick and 14 in. by 19 in. in size. Twenty of these plates, each having a precipitating surface of nearly 400 square inches, are placed in each compartment.

In the bottom of the tank the plates are placed upright and

* At Mansfeldt these residues are re-roasted if they contain as much as 0.023 per cent. of silver.

are slightly inclined, being separated from each other by a small strip of wood at the top, Fig. 38. Over this the plates are laid horizontally with strips of wood between each to prevent actual contact. This arrangement gives about 100,000 square inches of precipitating surface to each system. Both series of tanks are filled with copper in the same way. The tanks, while the precipitation is going on, are kept covered with wooden covers. At the end of a week they are removed and the copper plates shaken and washed in the liquid to remove the silver sponge which falls to the bottom and is taken out. This sponge is very light and adheres very slightly to the copper. After the copper plates are taken out the liquid is allowed to settle.

The copper solution is drawn into the tanks *n n*, and the silver carried to the tubs D to be washed to remove any traces of copper. It takes about two hours to get the tanks ready for another charge. More than half of the silver is deposited in the first four compartments. Here the copper plates last about four months. In the other compartments they last twelve months.

The amount of copper dissolved is equal to the quantity of sulphuric acid set free from the silver.

C. *Washing and Fusing the Cement Silver.*—The cement silver is washed in a washer invented by Professor Pearce and patented in England about the year 1866. It is a tub about 4 ft. high and 4 ft. in diameter at the top and 2 ft. at the bottom. It is, however, sometimes made a little smaller, being 42 in. high, 40 in. in diameter at the top, and 23 in. at the bottom. This tub, with its injector, is shown in detail at Figs. 39 and 40. Two of these washers are placed on a raised platform having a spout, connecting with the sulphate of copper tanks. About 3000 oz. of silver are placed in the false bottom of the tub. A mixture of one part of sulphuric acid to 100 parts of water is then placed in the tub in sufficient quantity to cover all the silver. Steam at a pressure of 50 lb. is then turned on through the injector and the arm moved so as to open the air holes. The steam and air pass down through the false bottom and up through the silver and sulphuric acid. A very violent ebullition is caused in the liquid by this passage of the air and steam.

The silver is thus kept in constant agitation, and fresh surfaces

are constantly exposed to the action of the acid. Besides this mechanical effect the current of air oxidizes the metallic copper and transforms it, together with the suboxide, into sulphate.* The cement silver from the tanks still contains some traces of copper as sulphate and some metallic copper detached from the plates.

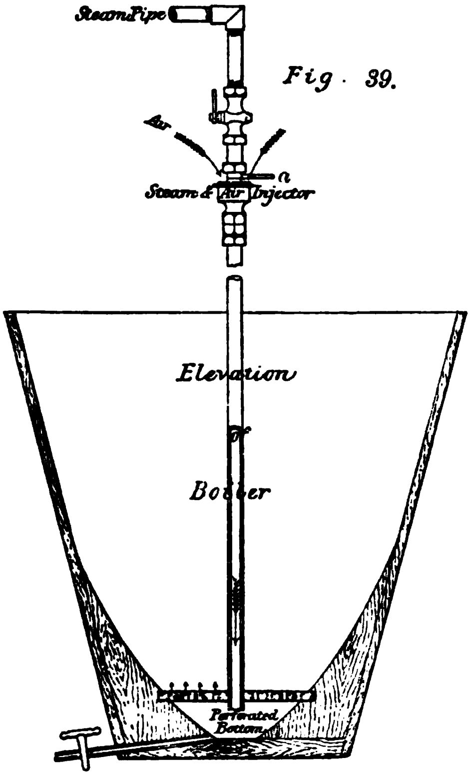


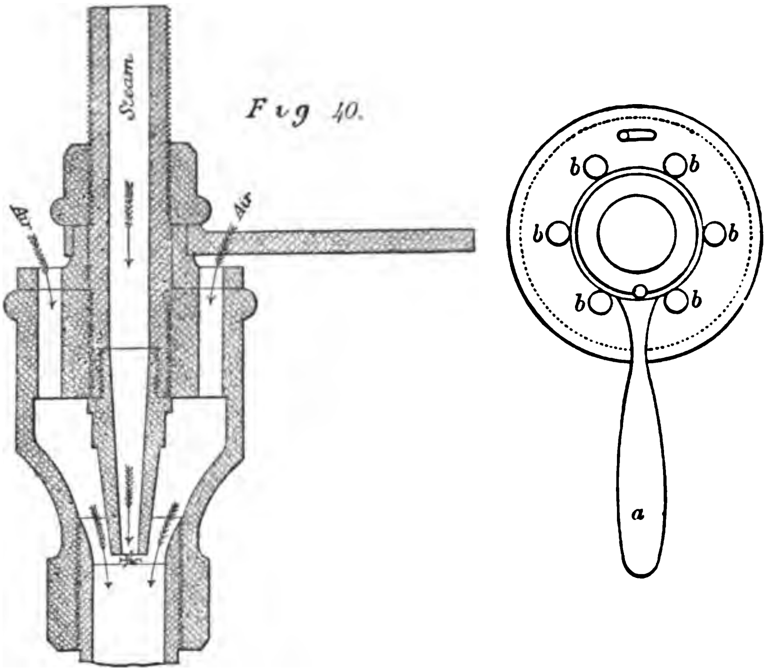
Fig. 39.

PEARCE'S WASHER.

At the end of two to three hours the liquid is run off through the spout into the tanks. The silver is then washed for half an hour with clean water and steam and then removed in buckets to be dried on top of the drying furnace, Figs. 41 and 42. It requires from three to three and a half hours to completely purify the 3000 oz. of cement silver. After drying, the silver is melted in graphite crucibles in the furnace, Figs. 43 to 45. It is cast in

* This injector is also used in parting the rich auriferous copper alloy, for the separation of gold, and manufacture of sulphate of copper.

Fig 40.



DETAILS OF PEARCE'S WASHER.

Fig. 41

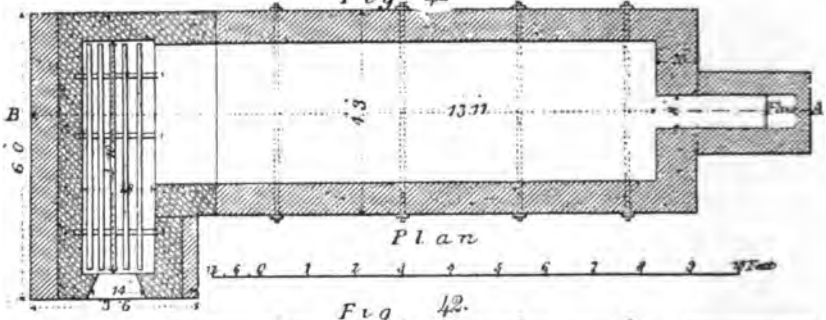
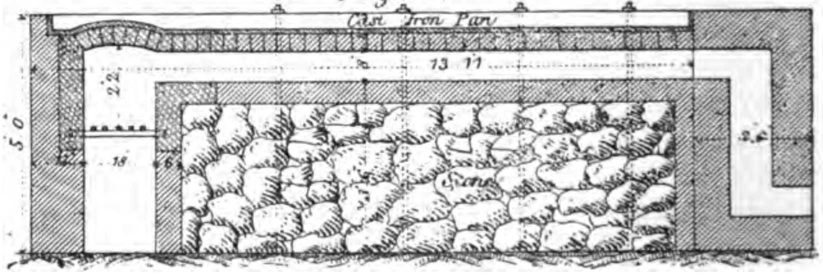
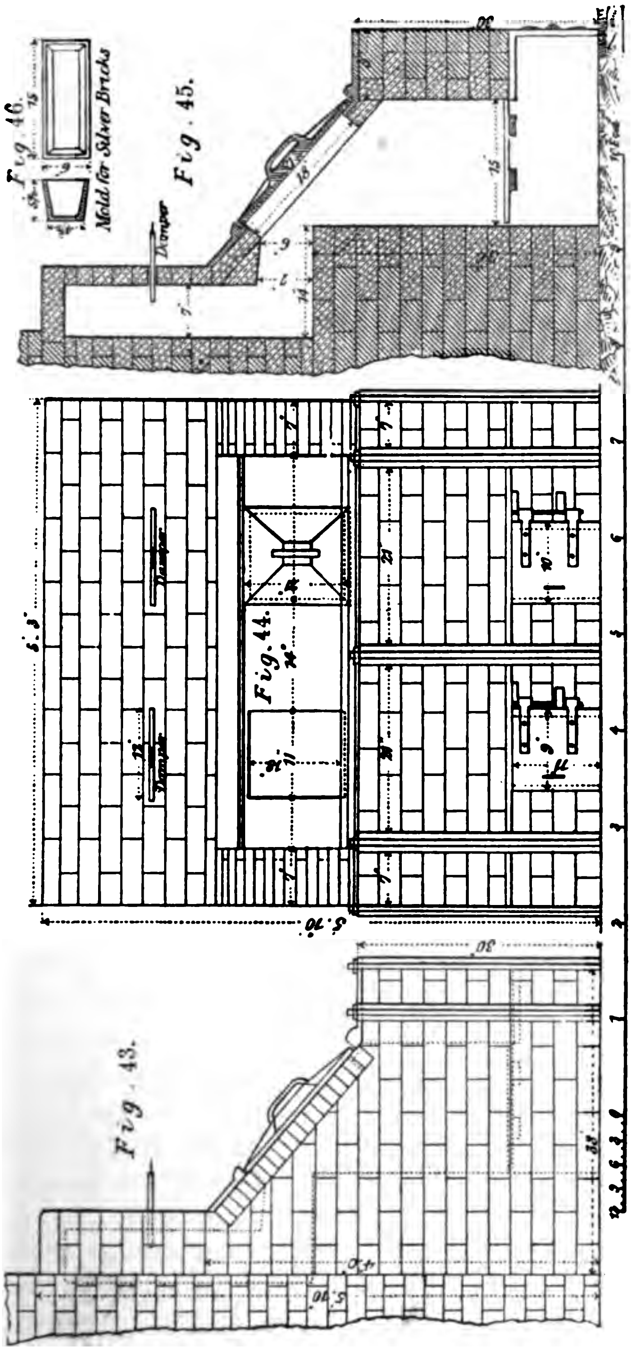


Fig 42.



DRYING FURNACE.



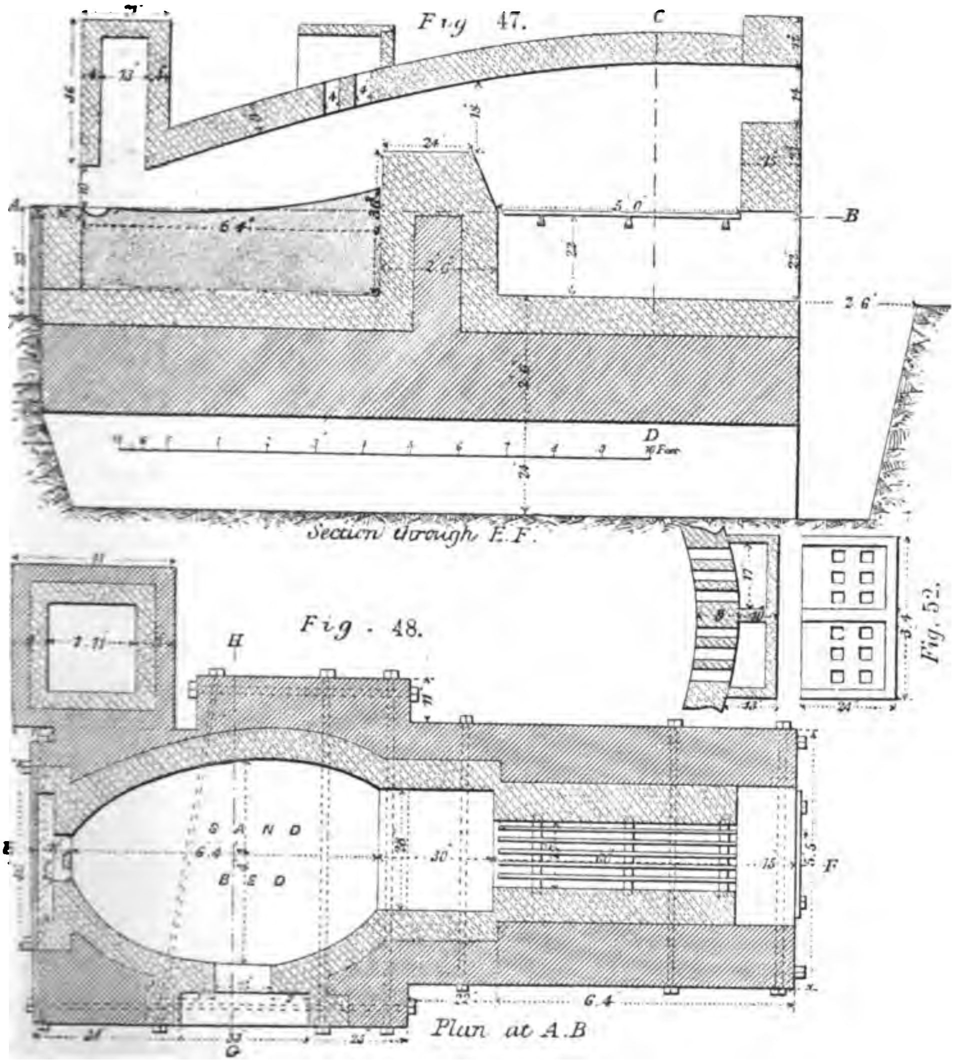
FURNACE FOR MELTING CEMENT SILVER.

heated iron moulds smeared with oil in the mould Fig. 46. The bricks are worth from $1200 to $1500, and are from 999 to 999.5 fine.

D. *Precipitation of the Copper.*—The copper solution from the tanks G, see general plan, Fig. 26, runs into one of the tanks *n*, Figs. 36 to 38, which are divided into compartments like the tanks *g*, and are also covered. These compartments are filled with scrap iron, which is simply thrown in without any special care in piling. The spent liquor, which is sulphate of iron, when sulphuretted hydrogen or a polished steel plate shows no trace of copper, is discharged into the stream ; the velocity of the discharge being regulated according as the action is quick or slow. The copper precipitates on the iron and is left to accumulate. The compartments are cleaned out about once a month. The copper is removed from the iron by simply moving it backward and forward in the liquid. The iron so cleaned is at once placed in an empty tank to be used on a fresh charge. All the iron used is old scrap iron, and is therefore not weighed. About 5000 lb. to 6000 lb. are used in each tank. The cement copper is allowed to drain and dry, and is then taken to the smelting furnace. It contains about 90 per cent. of metallic copper when it is fresh. The small amount of impurity is owing to the fact that the tanks are closed, thus preventing the precipitation of insoluble compounds of iron. The cement copper oxidises very rapidly in contact with the air, so that when ready for the furnace it does not contain more than 80 per cent. of copper in the metallic state.

E. *Refining the Copper.*—The furnace in which the cement copper is refined is shown at A in the ground plan, Fig. 35, and in detail by Figs. 47 to 52. The fireplace is 5 ft. long, and 28 in. wide at the top of grate. The grate has the same length, but is only 17 in. wide. The bridge is 2 ft. wide. The laboratory is 6 ft. 4 in. long and 4 ft. 4 in. wide, and has two doors, one at the end, which is the charging door, and one at the side, which is the working door. Just over the bridge, Figs. 47 and 52, there are two rows of six openings, each for the introduction of air ; these are covered with a hood to prevent the introduction of foreign substances. The furnace connects with the chimney by a flue, which is 2 ft. square.

The fireplace has 11.65 square feet and the laboratory 21.79 square feet, so that the relation between them is nearly 1 : 2. This furnace runs once a month for eighteen or nineteen hours. A charge of 2500 lb. of copper mixed with 50 lb. of refuse



charcoal is put in at 6 P.M. The fireman keeps up the fire during the night and the refiner takes it at 7 A.M., and then skims off the slag and exposes the surface of the bath. Considerable sulphurous acid is given off, probably from the reduction of the

sulphate of iron in the cement copper. The charge is worked for a "set," which takes three to four hours. This is done by striking the surface with the rabble and making waves. This is

Fig. 49.

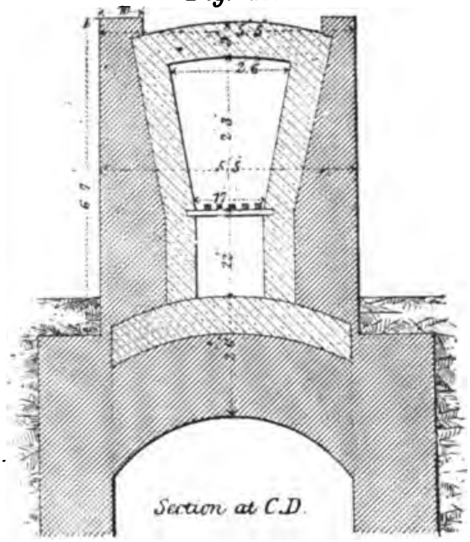
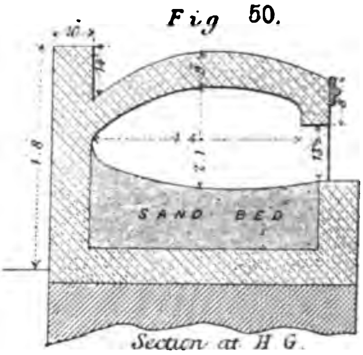


Fig 50.



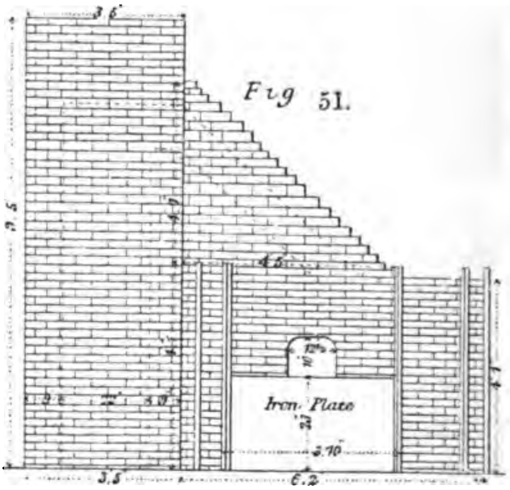


Fig 51.

called beating the copper. The copper produced contains from two to three per cent. oxide of copper dissolved in it, but it is not necessary to refine it completely, as it is used at once in the tanks

G. The copper is taken from the furnace with a ladle and is poured into a cast-iron mould made of a frame which is slightly tapering, being larger at the bottom than the top. This frame is placed on a cast-iron plate 3 in. in thickness. The ladleful of copper poured in is allowed to set, that is a film of suboxide of copper is allowed to form, another ladle is poured on, and so on until the mould is full. The cast-iron frame is then removed and the plates fall out separately as the oxide prevents anything more than contact. Twenty-five plates are made in this way at a time.

V. Treatment of the Ziervogel Tub Residues.

A. *Fusion for White Metal.*—The residues from the tubs consist of oxides of copper and iron with 20 oz. to 30 oz. of gold and 40 oz. of silver to the ton. They amount to about 22 tons a week. They are melted in the matte furnace, Figs. 28 to 31, with rich gold ores of the first class, containing iron with copper pyrites and variable quantities of gangue, and highly silicious tellurium ores. All the silicious pyritiferous ores are selected for this purpose. The ores are all crushed and put through a four-to-the-inch mesh sieve. The charge is brought to the furnace in alternate barrows of residues and ore, but it is not mixed before charging, as it becomes mixed after it is thrown into the furnace. The charge consists of:

	lb.
Tub residues...	4000
Raw gold ores of the first class	2500
Gold ores of the third class...	900
Total	7400

When there are no tellurium ores the charge of gold ores of the first class is made to amount to 3400 lb. The treatment is exactly the same as before. A poor slag containing only two ounces of silver and a trace of gold is produced; it is very much poorer than those of the previous fusion. It has otherwise very nearly the same composition as the others, but there is no zinc either as blende or oxide in it.

The matte contains:

Copper	60 per cent.
Gold	55 oz.
Silver...	130 ,,
Sulphur	30 per cent.

It is called white metal. If the matte was made richer in copper the slag would also be richer and there would be more loss. The tapping is made twice in twenty-four hours. In other respects the labour, fuel, &c., is the same as in the matte fusion No. 4. This fusion for the treatment of tub residues takes place once a month and lasts a week. All the plate slag produced during this operation is put directly back into the furnace.

B. *Roasting the White Metal.*—At the end of a week all the mattes produced are recharged in large lumps, the charge being about four tons. It is roasted at a dull red heat for about ten hours with admission of air. The reaction which takes place between the sulphide and oxide makes a peculiar noise which can be heard at some distance from the furnace. The operation is termed "roasting" for black copper, but it is stopped at half way. As the sulphur is driven off some metallic copper is liberated. The slag is very thick, and not more than 200 lb. to 300 lb. are produced. It contains from eight to ten per cent. of copper, and is highly basic, often containing crystals of magnetite. At the end of the ninth hour the doors are closed and the fire-place charged. The whole furnace is brought to a high heat so that the whole charge is in intimate fusion. Just before tapping it is rabbled for five minutes, and then tapped into sand moulds. The tapping is done as before, but moulds are made to receive the mattes as the charge is greater. In the first three or four pigs there will be found plates or bottoms of metallic copper containing arsenic, antimony, and lead. These bottoms contain nearly the whole of the gold, with from three to five per cent. silver, and eighty per cent. copper. The matte is pimple metal, and contains about:

Copper	75 per cent.
Gold	2 oz.
Silver...	120 ,,

From every charge about 600 lb. of bottoms, and 3 tons of matte, are produced. This bottom fusion takes three days, making ten days for this treatment of the residues. The labour is the same as in the matte fusion, but more wood is used, four cords being burned in twenty-four hours. Only two operations are made in twenty-four hours.

C. *Treatment of the Pimple Metal.*—The pimple metal is roasted again in the same way, treating it nearly five hours, and making four charges in twenty-four hours. Other bottoms are produced poorer in gold, but containing:

Gold	60 to 100 oz.
Silver...	300 oz.
Copper	75 per cent.
Sulphur	25 ,,

The pimple metal from this fusion contains:

Gold...	¼ oz.
Silver...	110 ,,
Copper	80 per cent.
Sulphur	20 ,,

the iron being entirely removed. This operation takes one and a half days. The bottoms are treated with the other bottoms. The pimple metal goes to the Ziervogel process B, but is kept entirely separate because it contains no gold as does that of the process A.

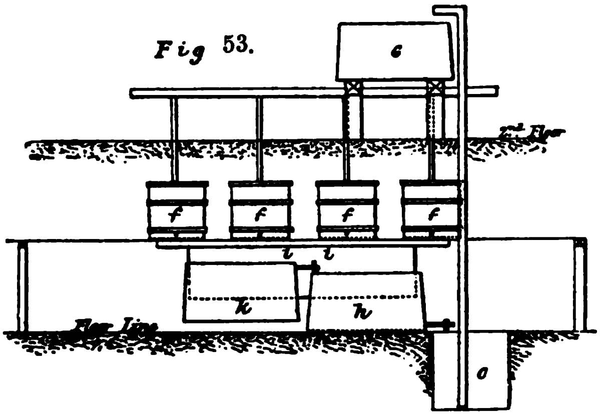
VI. Treatment of the Residues of the Ziervogel Process B by the Augustine Process.

The residues from the Ziervogel process B, which contain 25 oz. of silver per ton, are roasted with salt in one of the furnaces B, Fig. 35, for roasting for sulphate of silver in the Ziervogel process. The residues are charged moist, a charge being one ton. It is heated for two hours, until it is hot. Twenty pounds of salt are then added, and well rabbled into the charge for fifteen minutes.

The charge is then drawn to prevent the loss of copper, as well as chloride of copper. Three charges are made in twelve hours. This requires one man, and three-fourths of a cord of wood.

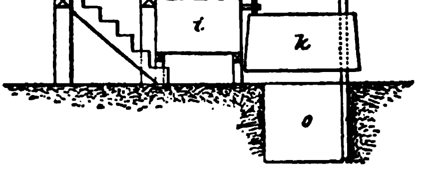
Solution.—This material is treated with a hot saturated solution of brine, a tank, holding 1000 gallons of the brine solution being always kept in reserve; 1600 lb. of the chlorurised residues are placed in a vat, and the solution allowed to constantly flow through it by an inch pipe for four hours.

Fig 53.



The liquid which runs out of the solution tubs runs into tanks, Figs. 53, 54, and 55, where the silver is precipitated with copper, and the copper with iron, as in the Ziervogel process. The salt solution contains chloride of iron, and is pumped back into the tanks, and is used again. Chloride of iron, by constant boiling,

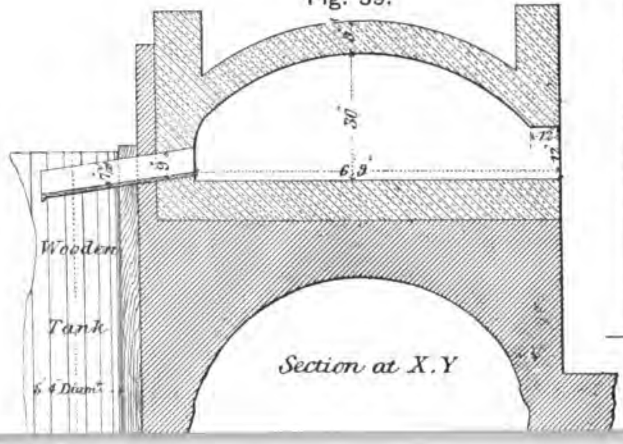
Fig. 54.



SOLUTION TANK

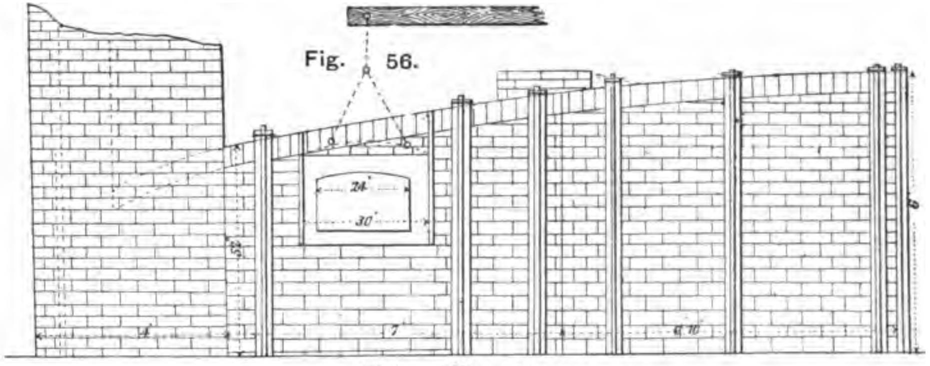
becomes perchloride, and finally sesquioxide, and is precipitated. The salt solution lasts, with occasional renewals of water, inde-

Fig. 59.



Section at X.Y

Fig. 56.

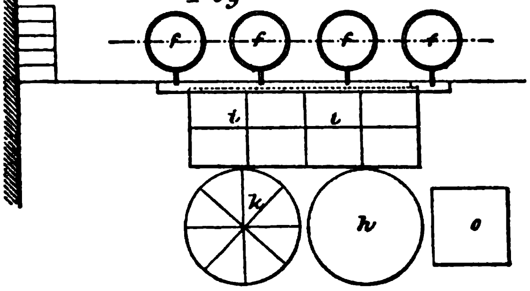


Side Elevation.

Fig. 57.

finitely. The loss of salt, per ton of residue treated, is about 10 lb. The residues from this treatment are either reduced and

Fig 55.



SOLUTION TANK.

made into ingots, or sold as they are as residues. The precipitation is the same as in the Ziervogel process, except that chlorides are formed. The material is always kept separate.

VII. Treatment of the Bottoms.

Four tons of white metal from the Ziervogel treatment, gives 600 lb. of bottoms. These are left to accumulate until they amount to 3500 lb., enough for a charge in the small reverberatory furnace. The furnace in which this operation is effected is shown in the general plan at C, and in detail in Figs. 56 to 61. The fireplace is 6 ft. long, 4 ft. deep, 42 in. wide at the bridge, and 20 in. at the grate. The bridge is 4 ft. wide. The laboratory is 9 ft. long, 6 ft. 9 in. wide, and connects with the chimney 2 ft. 6 in. square, by a flue. The surface of the fireplace is 21 square feet, that of the laboratory 46.27 square feet; the relation, therefore, is 1 : 4. The furnace has a working door at the side and a charging door at the end. On the side opposite the working door there is a spout which ends in a wooden tank sunk in the ground, which is 4 ft. 5 in. in diameter and 3 ft. deep. The object of the process is to oxidize the lead and other impurities, and to prepare the metal for treatment for gold. The charge is made at 7 A.M. It is first sweated at a low temperature for two or three hours, during which time some of the lead liquates and runs out of the furnace. It is then left to oxidize for three or four hours. In about seven hours the charge is well melted. The slag, which is

skimmed at this time, is composed mostly of oxides of lead and copper containing from 10 to 15 per cent. of copper, and is sent to operation No. 3. After the slag is withdrawn, the bath is beaten with a rabble for about two hours, all the doors being opened to admit an excess of air. It is again skimmed and tapped into water. The "pitch," that is the condition of the copper, must be such that the whole of the sulphur is eliminated before the oxygen is absorbed. If the pitch is right, the globules will all be round and hollow. This point must be seized with the greatest nicety, for if the charge remains too long in the furnace the globules will cast solid, and the charge must then be put back and worked with sulphur. The temperature of the water governs the size of the globules. They are small when it is cold and large when it is hot, but it does not otherwise affect it. It takes about ten minutes to do this casting. The copper flowing from the spout falls on to a pole of green wood held underneath it, so as to scatter the copper. Care must be taken that the slag does not flow with the copper. To prevent it the doors are opened, so that the slag is cooled until it is pasty. One charge is made at a time, and only one or two per month. The globules contain 1000 oz. of gold, 600 oz. of silver, and a trace of lead. Twenty tons of white metal give one ton of refined auriferous copper. Three cords of wood are used, one man tends the furnace, one man does the firing.

VIII. Treatment of the Oxidized Copper Alloy.

The copper globules are oxidized in one of the fine calciners, in which sulphate of silver is treated. One and a half tons are charged at a time. The oxidation takes thirty-six hours. The globules are put into the furnace in a heap and spread out over the hearth. The charge will be 3 in. deep. The fireplace is charged at once, and the temperature is made as hot as the red bricks will bear, and as oxidizing as possible. It is constantly rabbled. At the end of thirty-six hours a portion is taken out and tested, to see that it will pulverize completely. If it does the operation is finished, if it does not the oxidation is continued. The whole of the copper

has been transformed by the operation into suboxide, and the charge is increased in weight about 500 lb. by the operation. The grains are black on the outside, but if broken or rubbed the streak is red The charge is drawn out into an iron barrow and carried to the store-room. It is placed in bags, packed in petroleum casks, and shipped to Boston. One cask holds 650 lb. Three cords of wood are used for the process, and two men do the work, one man to each twelve hours' shift. The men are required to bring their own wood.

The oxidized product is treated with dilute sulphuric acid. This is done in a conical tub lined with lead with a false bottom. The bottom is hollowed so as to leave as little space as possible. A charge is 1500 lb.; over this sulphuric acid at 20 deg. Baume is poured. Steam and air are turned on and the boiling continued for four hours. The whole is not dissolved, but 90 per cent. of the copper will be in solution. It is allowed to settle for an hour, and is syphoned off and a fresh charge put in. Two charges are made in a day. This is repeated until all the oxidized products have been treated. This work is not done at night. The residues are boiled two or three times in the same way to get out all the copper possible. The tub is then cleaned up and what remains is melted in plumbago crucibles. The bullion is from 600 to 800 fine of mixed metals. It contains from 40 to 50 per cent. gold and 20 to 30 per cent of silver. This is sent to the mint.

The sulphate of copper is crystallized and sold. The mother liquor is used to dilute the acid used for the solution of the oxides.

The working of these alloys of gold, silver, and copper was first tried in the works, and was given up on account of the high price of sulphuric acid. It was carried on for more than a year in Boston, but has quite recently been abandoned, and the separation of gold and silver is now to be done at the works by a process invented by Professor Pearce.

CHAPTER IV.

CRUSHING MACHINERY.

CALIFORNIA STAMP MILLS.

THE peculiar conditions under which the processes for the extraction of gold and silver west of the Rocky Mountains exist, have developed in these regions processes which are characteristic of the district where they are used. They have been invented under special exigencies and difficult conditions which characterise all newly settled regions remote from the centres of civilisation, and which are never the same in different districts. Attracted by the rumour of untold wealth easily accumulated, and by circumstantial stories of suddenly made fortunes, the miner in the early days found himself in the "diggings" in the face of stern necessity obliged to continue his work with low-grade ores, difficult to mine, and rebellious to treatment. The result of the effort of the miner in the Western States and Territories to work out the problem of living has been that a number of machines, many mechanical devices, and a series of processes have grown up, little by little, in the course of years, to which the unsuccessful have contributed their experience and the successful their energy and capital, until the ores of the Western States are now worked on carefully elaborated processes, which either no one person has invented, or the inventors of which have been forgotten.

The processes used in Europe both for crushing and for treating the precious metals, were either unknown, or were based on saving what would not pay to work in this country; involving in many cases, at the same time, an outlay of capital which, if the miner had possessed, he never would have brought to the "diggings" to invest. It was not of so much consequence whether he obtained all the pay, or even a large percentage of it, as that he should obtain *some* of it. All of his ore had to be crushed in order to bring the precious metals in contact with mercury. And thus, starting with the rock-breaker, Fig. 1, which he found the inhabitants of the country using when he went there, then sub-

stituting for it the Chilian mill and the arastra of the Mexican patio, or the pan which was his household utensil and implement of labour at the same time, the system of hydraulic mining grew up for treating poor placer gold, and the California stamp mill with its amalgamated plates, pan, and blanket processes for treating free-milling gold and silver ores on a large scale. The early miner had no time and no inclination to stop to make nice distinctions. He knew that quartz contained gold and silver, and hence any and every rock containing the precious metals, no matter what it was, was to him quartz. With him the important question was, not what its composition was, but how he could readily get the pay out of it.

The California stamp mill is, undoubtedly, the legitimate growth of the old German stamp with its clumsy working, without considering time as an element of value. The high prices of everything necessary to life have, however, developed out of the old inefficient wooden battery a machine of such perfection that beside it the Hungarian stamps of to-day seem always to have been clumsy, cumbersome, and wasteful in the use of power; and we are apt to forget, in the face of new machinery, the real service they have rendered in former times. The Mexican patio plan of amalgamation, adopted from the original settlers and the barrel process, used by the early miners, have produced the pan with its machinery and appliances, which is so different from either of them, that we accept the fact of their growth as a matter of history, although hardly able to trace any of the steps of it. So distinctive, however, are all these methods, that they are known wherever the precious metals are worked, as the American processes. At first rebellious ores were not treated. Only free-milling ores were stamped, but when it was found that the rebellious were often richer than the free-milling ones, they were roasted in heaps previous to crushing.

The California stamp mill is such an important element in the working of both gold and silver vein matter, that before describing the direct or indirect amalgamation of either of these metals, it must be discussed in detail. It consists of several distinct parts which are adjuncts to it, serving to hold it together or to make it efficient. The stamp itself is made up of the stem,

the tappet, the boss or head, and the shoe; it works on the die in the mortar. The framework by which these are supported carries the cam-shaft and the cam, and behind them are the bins for receiving the ore which it is to treat.

The California stamp differs from the ordinary European stamp in the fact that all its parts are interchangeable, and are made either of iron or cast iron or steel; and from the ordinary Cornish stamps in that the stems, shoes, and dies are made round, and that, by an arrangement of the tappet and cam, the whole stamp is made to revolve a certain fraction of a circle at every stroke of the cam. There are many other minor details, but these are the characteristic ones. They are arranged both for wet and dry crushing, and differ somewhat in detail, according as they are used for one or the other. We shall discuss the stamp in the order of the foundations, the frame, the mortar, the die, the cam, and the cam-shaft, and the stamp proper, giving the differences as the mill is arranged for gold or for silver, or for wet and dry crushing, as each special principle is described in detail. These differences pertain mostly to the mortar and screens, and not so much to the stamp itself. The foundations are the most important part of the construction, as the efficiency of the battery depends mainly upon them. If they have been properly prepared, the effective horse-power of the battery will be large. Whenever they are improperly made, from a desire to do the work cheaply or from carelessness, the batteries cannot run at a high speed without racking themselves to pieces, but if properly constructed they can be run for a great length of time without breaking a bolt. There are a number of very expensive mills in California and Nevada, where a considerable part of the power of the battery is expended in simply shaking the structure to pieces, owing to the improper construction of the foundations, and anchorage of the battery blocks and frame. The greater the care given to this part of the structure the less the repair account will be, and the greater the amount of effective work done by the stamp.

For the foundations the earth must be removed to the bed rock which must be carefully trimmed and levelled. The depth of the trench will vary from 6 ft. to 14 ft. according to the nature of

the ground. A wall of masonry, 6 ft. thick, is then built on both sides of it, as at the Keystone Mill, making a trench, 8 ft. wide at the top and 6 ft. at the bottom, the whole length of the battery. At the Consolidated Virginia all of this trench was cut out of trachyte. The bottom was levelled up, and pounded with an iron stamp to make it firm. Cement was used to make an

Fig. 62.

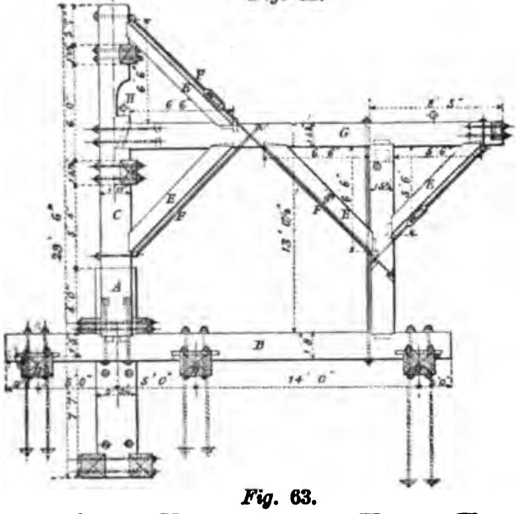
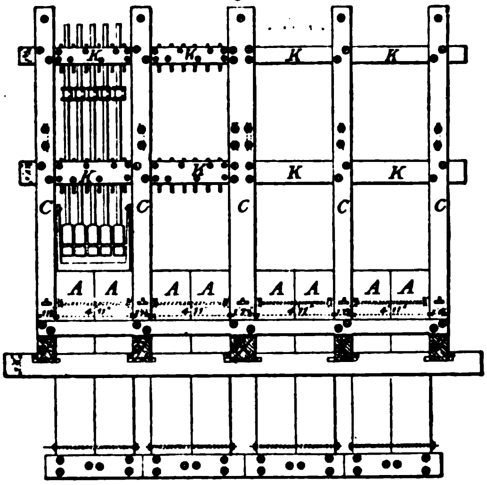


Fig. 63.



STAMP MILL FRAMING.

accurate floor. The pit was thus made large enough to leave a space of 2 ft. all around the mortar block, which is generally filled with poor battery sand.

At the Consolidated Mine, the mortar blocks A, Figs. 62, 63, and 64, are 30 in. by 30 in. square, and 12 ft. to 14 ft. long. They are

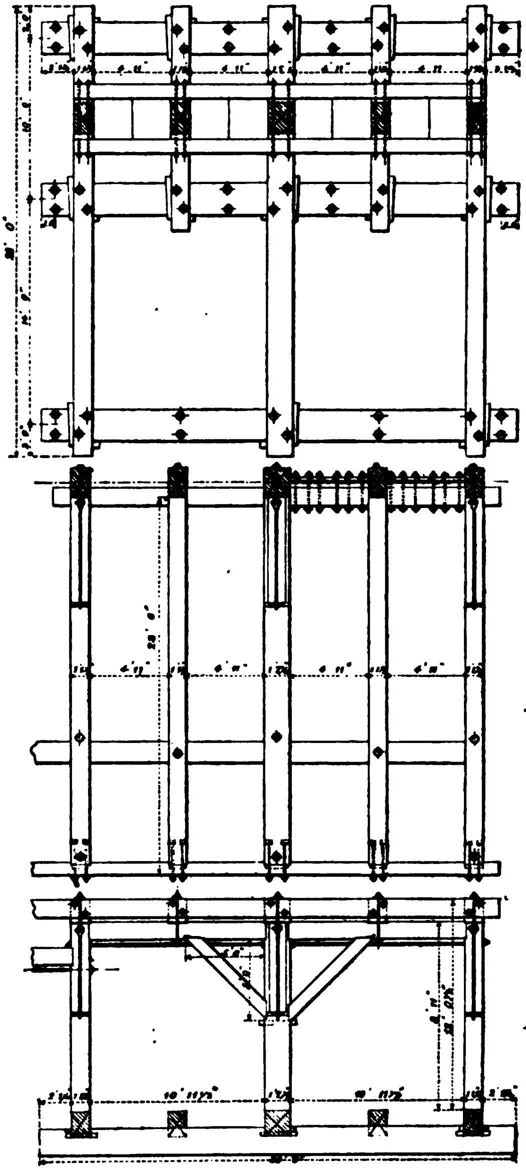


Fig. 64.

Fig. 65.

Fig. 66.

STAMP MILL FRAMING.

made perfectly true, and both sides are coated thoroughly with hot Stockholm tar, so as to have an excess of it, and to fill up all

the cracks. They are then bolted together by six bolts, $1\frac{1}{4}$ in. in diameter, the nuts of which are screwed down by two men with a 4 ft. wrench. The foot timbers for the mortar blocks are 18 in. square and 6 ft. long. They are let 6 in. into the mortar block, and are freely tarred and bolted together with six $1\frac{1}{8}$ in. bolts in the same way as the mortar blocks, having been first accurately squared. The bottom of these blocks with the mortar blocks in the centre, forms its support. Five feet from the top the mortar blocks are sized 59 in. by 29 in. The blocks so prepared are let down upon the floor, and once there, if there has been any want of accuracy in the level, sand is thrown in to level up with. Once in place, the height of the block is determined accurately, and sighted through the whole line on a level. Boards are then nailed on this line, which is usually 2 ft. to 3 ft. from the top, and the projecting part sawed off. The top is then planed smooth, making it about $\frac{1}{16}$ in. hollow, to insure the block not becoming rounded. In the interval before the mortar is put on, the top is covered with boards to secure it from injury.

The mortar block once in position, the frame, which is generally made of the best red spruce or sugar pine, must be raised. For this purpose three battery sills, D, Figs. 62 to 69, 18 in.

Fig. 67.

Fig. 68.

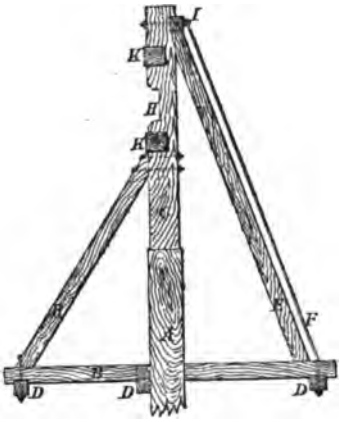


Fig. 69.

by 24 in. and 28 ft. long, are placed in position parallel to the direction of the cam-shaft. The distance between the centres

of the first two, one in front and one behind the mortar block, and that of the block, is 5 ft. The other is 14 ft. from centre to centre in front of the first sill. These sills are secured by bolts 8 ft. long, which are keyed either into the masonry, or to the bed rock. In case the sills are to be anchored to the bed rock, holes at least 3 ft. deep and 1½ in. in diameter are drilled into the rock. The bottom of the bolts which are to secure the sill are slotted 6 in. from the end, and wrought-iron wedges ⅝ in. by 1 in., and 5 in. long, with a head 1 in. square, are made to fit the slot; the bolt is placed in the hole and driven so that the wedge enters up to its head. The holes are then filled with melted sulphur. The bolts must be long enough to admit of cast-iron washers and nuts on top of the sills. The outside line timbers, B, Figs. 62, and 67 to 69, for each battery, 14 in. by 20 in., and 28 ft. long, are now placed in position. These are wedged into the battery sills, and securely fastened by two bolts in each sill. The top of the sill is 4 ft. below the top of the mortar block. The centre line timbers, Fig. 64, are 20 in. by 20 in. and 28 ft. long. The intermediate line timbers are 14 in. by 20 in. and 28 ft. long; they are dressed on the top side and sized to 13½ in. and 19½ in. each, where they pass the battery blocks. They are let into the sills at least 3 in., and secured by keys driven both ways, and by two iron bolts 33 in. long and 1⅛ in. in diameter. The outside battery post is 23 in. by 13½ in., and is tarred and let into the sills. The posts for four batteries are raised at once; as the middle one of the five has to bear a greater strain than the others, it is made 23 in. by 19½ in. They are secured to the line timbers by two 1-in. joint bolts, each 44 in. long. The rest of the frame is shown in Figs. 62 to 66. In the upper part of these posts the cam-shaft journal seat is cut, H, Figs. 62, and 67 to 69. The posts are held together by the stamp guides K, the lower one of which is 17½ in. by 13½ in., and the upper one 13½ in. square. The method of supporting these posts against strain is shown in Figs. 62, and 67 to 69. Fig. 62 is that of the Consolidated Virginia Wet Crushing Silver Mill; Fig. 68 is a wet crushing gold mill; Fig. 69 is a dry crushing silver mill; Fig. 67, the old form used for both wet and dry crushing; Figs. 67 and 69 are being rapidly abandoned; they allow the whole battery to spring, as they are hung

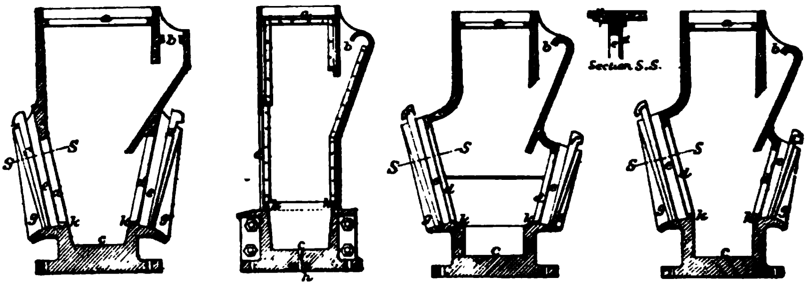
by means of iron rods; and both are objectionable, because they are braced on the feed side, where it is desirable to have the greatest freedom of passage. The object of the bracing should be to allow of the greatest speed of the heaviest stamp, with the least possible shaking of the timbers. Any outlay of money to achieve this purpose will very shortly pay for itself in the repair account, and no amount of repair will remedy defects of original construction. As the forward strain, against which the battery must be braced, is the pull of the belt which runs the stamp, no bracing of any kind is needed on the feed side, if the battery is properly constructed. The form, Fig. 62, which is that of the Plumas Eureka Gold and Consolidated Virginia Silver Mill, is the best and strongest construction, and is very generally adopted as the most suitable for heavy stamps. A travelling Weston pulley block and tackle is usually suspended over the stamps so that it can be moved along the whole line of the batteries as soon as the frame is finished. It is hung from the roof girders by wooden hangers, and secured to them by $\frac{3}{4}$ in. bolts. The track, which is 1 ft. wide, is furnished with iron rails, on which cars run which are fitted with eye-bolts for attaching the chain blocks. One set of chain blocks for every four batteries, or twenty stamps, are usually provided. This is done to facilitate the operation of putting up the stamp itself, as well as for changes and repairs afterwards. When the frame is ready the mortar is placed in its position on the mortar block.

The holes for the mortar are bored from the template taken from the bottom of the mortar. When the mortar is to be put in, the temporary covering on the mortar block is removed. The season cracks are filled up with sulphur, the block again planed, carefully coated with tar, and covered with three thicknesses of common house blankets, costing $9 per pair, which are carefully coated with tar on both sides. The mortar is placed upon these blankets and bolted with $1\frac{1}{2}$ in. bolts. If the mortars were bolted directly to the blocks they would after a time get loose, and sand would work between the mortar bottom and the block, and the mortar be thrown out of plumb by its introduction beneath it. If sufficient care is taken in the con-

struction of the mortar and the mortar blocks, the battery is almost free from jar.

The mortars, Figs. 70 to 73, are all made of cast iron, but

Fig. 70. *Fig.* 71. *Fig.* 72. *Fig.* 74. *Fig.* 73.



STAMP MILL MORTARS.

differ in construction, according as they are to be used for wet or dry crushing, or for gold or silver ores. The dry crushing mortar, Fig. 70, is the same for crushing both metals. For wet crushing each metal has its own peculiar form. There are a great many patterns of these mortars, but only the best types will be described, such as are used in the most recently constructed mills. Figs. 70 to 73 give the various kinds of mortars generally made by the foundries of San Francisco and in the East. They are always cast solid when they are to be used in districts easily accessible, but when they have to be transported by wagons they are made in parts and bolted together. They weigh from 3000 lb. to 6000 lb., according to the pattern. Those at the Keystone Consolidated Gold Mill in Amador County, California, weigh 5400 lb. They are always cast heavy on the bottom, as it is here that there is the greatest strain. The sides of mortars used in districts remote from foundries, are generally cast thin, and are protected with repair piece linings which are constantly replaced. They are, however, sometimes cast extra thick to be used without linings. The peculiar advantage of the California mortar in all its forms is its strength and completeness, as all the parts which can be, are cast in one piece, and it therefore can be set up or replaced when necessary with the least possible delay. It is very durable, since extra strength is given to all the parts likely to give out, and all those exposed to wear are specially protected, particularly those which are liable to be broken. They are

usually about 4 ft. 7 in. long, and from 4 ft. 2 in. to 4 ft. 4 in. high, and 12 in. wide on the inside, where the dies are set. The bottom varies from 3 in. to 6 in. in thickness, and has a heavy flange cast on it, so as to allow of its being bolted to the battery blocks, to which they are secured by nine ⅞ in. bolts. The dry mortar, Fig. 70, is cast in one piece, or is made sectional, like Fig. 71, when it is to be transported. The bottom of this latter mortar is made of cast iron, cast in transverse sections, into which a slot *h* is carefully planed out for the purpose of fitting a wrought-iron bar into it to which the sections are rivetted by very strong bolts turned to a size, and driven after a reamer. This holds it securely together. The upper part is made of boiler-plate held by angle-iron. When set up it is perfectly firm. When made in one piece the die is set high. The screens are slightly more inclined from the perpendicular, to admit of a more easy discharge. Sometimes the discharge is made from one, and sometimes from both sides. It would seem that the delivery would be greater with a double than with a single discharge, but the authorities are not agreed as to whether the single or double discharge is the more advantageous. The width at the bottom to which the die is set is about 11 in., and the whole mortar is 52 in. on the outside. In wet crushing, Figs. 72 and 73, the two mortars differ considerably, though they are in many respects alike. The height of both is 4 ft. 4 in., the bottom is 3 in. to 6 in. thick, and in most cases even thicker. The sides are from 1½ in. to 1¾ in. thick. The flange at the bottom for fastening it to the mortar block is 2½ in. thick and 4 in. wide. The gold mortar, Fig. 72, is of greater capacity than the others, and is adapted to receive, besides the ordinary lining, a complete lining of copper plates. It is necessary to have the most convenient arrangement possible for placing and removing these plates, and for securely fastening them to their position; as the removing and cleaning them is the most important work about the mill, and requires frequent stoppages, which should not be longer than can possibly be avoided. At the lower part of the discharge screen a step *k* is arranged for a copper plate, for it is at this point more than at any other that the coarse gold settles and should be caught before it has been either ground up or

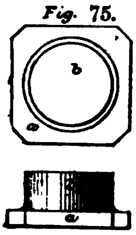
stamped into float leaf. In order to prevent the too rapid wear of the sides at the bottom from the constant splash of the pulp, a lining of hard cast-iron plates about 1 in. thick and 24 in. high, is placed around the bottom and bevelled at the ends so as to fit tightly against each other where they meet. They are held on the bottom by the footplate of the die, and so secured do not need any other fastening. They last from three to six months, depending on the kind of rock stamped. The feed slot *b* is on one side a little below the top of the mortar. The side here generally turns over and forms a catch against the splash which might throw some of the pulp out of the mortar. A sort of pocket is thus formed between the feed slot and the side which is always kept full of ore. The feed lining extends down into the battery beyond the main wall. Between it and the mortar wall there is an opening from 3 in. to 4 in. wide, so as to restrict the size of the rock, through which the ore is discharged upon the die and under the shoe. As the discharge is double, and from 12 in. to 18 in. high, a slot shown in section, Fig. 74, is placed at *e* to receive the screen frames, which are fastened to their places by iron wedges *f*. These are so made that they can easily be removed to replace a broken screen. The splash-box, which is made of wood or of cast iron, and has three or more spouts to distribute the ore over the blankets or amalgamated plates, is bolted to the mortar. The top of the mortar *a* is covered over by two 3 in. planks held in place by bolts. They join in the centre, and have holes cut for the passage of the stem.

The wet mortar for silver, Fig. 73, does not differ essentially from that for gold. As it does not receive the copper plates, it is a little less voluminous. The die seat C is generally higher, as it is not planed off at *k*, and the pulp discharge 2 in. to 3 in. above the top of the die. As the principal object is to discharge the pulp as soon as possible, the discharge screens are brought nearer to the middle line of the stamp, and have their entire surfaces available.

The mortar being in place, the dies are put into it, and as the die-foot is bevelled, it simply requires to be driven into the bottom of the mortar. The die, Fig. 75, is made of hard, tough cast iron, and consists of two parts, the footplate *a* and the die

proper, or boss *b*. The footplate is square with the corners cut off. It is usually 1½ in. to 2 in. thick, and from 10 in. to 12 in. square. At the Lincoln and Stanford Mills it is 11¾ in. by 9¾ in. The die proper is cylindrical, and is from 3 in. to 5½ in. high, and from 8 in. to 10 in. in diameter. It is chilled down to the footplate, and is replaced when it is worn down to that point. The footplate is made square, so as to fit the recess made for it in the mortar; if it were made round, the bottom of the mortar would not wear evenly, and might be worn through in places while it was still good in others. When it is square, it fills the bottom of the mortar, and furnishes the open spaces at the corners by which to pry it out when it is worn down. In order to prevent the wear of the bottom of the mortar, 2 in. to 3 in. of sand is sometimes put in the die seat as a bed for the dies to rest on. A great deal of difficulty has been found in keeping the dies in their places in dry crushing; there is no difficulty in wet crushing. For dry crushing the bottom of the mortar is sometimes filled with lead, which works very well.

Fig. 75.



In Lake Superior, and generally where Cornishmen are in charge, it is the custom to fill the whole mortar with a single die, which is turned over when it commences to wear unevenly. This can only be done when rectangular shoes and dies are used, but the iron never wears evenly unless the stamp is a very light one. The die is also apt to break before it is worn out, and no such practice is usual in California.

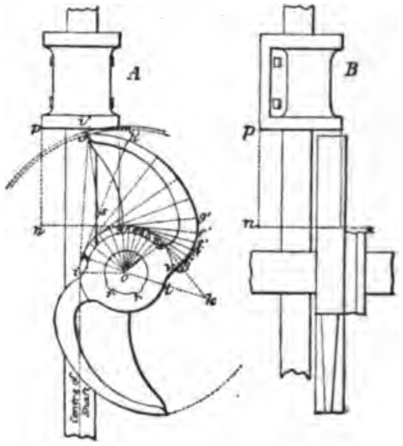
The cam-shaft with its cams is now put in place. The places for the journals of the shaft are cut in the battery posts at H, Figs. 62, and 67 to 69. The cam-shaft is of wrought iron. When used in a battery of five stamps it is generally 4½ in. to 5 in. in diameter, and when used for a battery of ten stamps at least 5 in. The tendency is rather to increase than to diminish these dimensions. The shaft for two batteries is 14 ft. 8 in. long. It is carefully turned true in a lathe, and has generally two key seats *r*, Fig. 76, 1 in. by 3¼ in., cut in at right angles to each other, to insure that the cam shall be normal to the shaft. It is fitted with a pair of cast-iron pulley flanges, ¾ in. at the outer rim, and 1¼ in. at the hub, which are faced in a lathe. Each one is fitted

to the shaft by a steel key. The bolt holes for fastening the pulley are counterbored, so that the bolt-heads will be flush with the face. The bolts are $\frac{7}{8}$ in. in diameter. The shaft is also fitted with two sets of collars.

The centre of the cam-shaft is 5 in. to 10 in. from the centre of the stem, and 9 ft. to 10 ft. from the mortar bed. The bearings rest on supports attached to the frame posts, on the discharge side of the battery. In wet crushing they are sometimes protected with Babbitt metal, but none can be used in these shaft boxes, in dry crushing, on account of the dust, as it cannot be kept smooth; it is therefore run in smooth iron boxes. It was formerly the practice to have a single cam-shaft to drive all the batteries of a mill, but it is now generally considered better to multiply the shafts, so that a pair of batteries, or each battery of five stamps, has its own. Each battery or pair of batteries will in this way have its own driving wheel. By this disposition, when one battery is out of order, the whole mill is not obliged to stop.

The cams, Fig. 76, are made of the best cast iron, and are T-shaped. They are 2 in. to 3 in. wide on the working face,

Fig. 76.



STAMP MILL CAMS.

and 2 in. thick. In order to prevent too rapid wear, the face is chilled.

The rib is $1\frac{1}{4}$ in. at the toe, increasing gradually until it is $2\frac{1}{2}$ in. at the hub. The hub is always strengthened by a band of the best

wrought iron, $\frac{5}{8}$ in. thick, and $2\frac{1}{2}$ in. wide, which is shrunk upon it. Single, double, and triple cams on the same shaft have been used; the usual form is a double cam, which is preferred, since it gives two drops of the stamps for every revolution of the shaft; it saves friction, and allows the battery to be run at a high velocity without increasing the speed of the engine. Three cams have not been found to work well, since they decrease the height of the fall of the stamp. The double cams were formerly cast in two pieces, and were screwed together, as it was supposed that this would facilitate the replacement of those which were worn out or broken. This was found, however, to be no great advantage, as they were always getting loose. They are now cast double in a single piece, being diametrically opposite on the same hub, which is always strengthened. The European plan of having a large number of cams on a long cylinder has been entirely abandoned. The hub projects beyond the edge and is always placed on the side away from the stem, so that the cam and the cam-shaft are as close as possible to the stem.

A cam curve is an involute of a circle, slightly modified at the end, as is shown in A, Fig. 76, the radius of which is equal to the distance from the centre of the cam-shaft to the centre of the stem. As the point of contact between the cam and tappet must always be on the line of the centre of the stem, an involute is the curve which will produce this result. The curve is usually described by cutting out a circle of a thin board *o i*, having this radius. At a point on the circumference at *i*, a string is attached, the length of which is determined by the lift which it has been decided the stamp should have; and to the other end a pencil point is attached. The circle with the thread wound on its circumference is laid upon a smooth board, and the string, being kept tight, is unwound, commencing at *v*, until it forms a tangent to the circle at the point where the other end is fixed. The dotted lines give the position of this curve at every point. This gives the curve, which is a complete involute. It is always modified at one, and generally at both ends, so as to stop the upward motion gradually, and to allow the cam to act on the tappet at the least possible distance from the cam-shaft. To

effect this, a point is taken 4 in. from the toe of the cam, and a circle is struck from the centre of the cam-shaft, thus reducing the lift of the cam; or by taking some point *s* on the tangent *i h*, and with a radius *s i*, striking an arc of a circle. This modifies the last 4 in. of the curve, dropping it ⅜ in., and, by allowing the least possible concussion on the tappet, diminishes the wear of the toe of the cam as the stamp falls. This form of cam admits of its receiving the weight of the stamp as near as possible to the centre of the cam-shaft, when the lifting motion is slow, and when the concussion is reduced to a minimum. The change in the curve at the toe prevents its too rapid motion on the face of the tappet. As the curve of the cam is determined by the distance between the centres of the stem and cam-shaft, this distance must be carefully adhered to in setting up the stamp. To get the position of a double-toed cam on the shaft its semi-circumference is divided into ten, and for a single-toed cam the whole circumference is so divided. When the distance between the centres is 5½ in., the drop will be 10 in., and it will be possible to make eighty-five drops per minute. When ninety drops are required with the same cam, the fall will be 8 in., and whenever higher speed is to be attained the drop is correspondingly reduced. When a very high velocity is required, a single-toed cam is preferred, the involute running two-thirds of the circle of revolution. The drop or lift varies generally between 8 in. and 11 in.; it is determined necessarily by the length of the cam. It is usually about 10 in., and the length of the cam corresponding to this drop is 21½ in.

As all parts of the cam are round, the upward motion of the cam tends to turn it by its friction against the tappet, so that the same face will not strike the place where the ore is in the mortar in exactly the same way twice in succession. The result will be that, unlike the Cornish stamp, both shoes and dies will be uniformly worn, as there is a constant rotating motion on the die. The experience, however, of the Quincy and Central Mine in Lake Superior has shown that the importance of this has been somewhat overestimated. There they prefer these rectangular shoes, dies, and stems, and assert that the result of their ex-

perience is in favour of the shape of the old Cornish stamp. The experience, however, on Lake Superior with the California stamp is very limited.

When the cam-shafts and cams are in position, the stamp proper is ready to be set up. This consists of the stem, the tappet, the head or boss, and the shoe. The stem is made of wrought iron carefully turned in a lathe throughout its whole length. It varies from 2⅞ in. to 3½ in. in diameter. It is 12 ft. to 15 ft. long, and at each end is turned tapering for a distance of 6 in. to 8 in., at the rate of 1 in. in 20 in., so as to fit into a tapering hole in the boss or head. As it is liable to break at this point, both ends are turned off, so that when broken at one end the other can be used before it is repaired. It weighs from 300 lb. to 450 lb., and is by far the heaviest part of the stamp. There is a decided advantage in thus putting a very considerable portion of the weight in the stem. Those of small diameter are liable to spring and bend from the blow of the cam, and to wear both the guides and cam rapidly; and if left too long a time, one stamp might work against the other. They are set in the guide boxes about 8½ in. apart.

A Table below gives the dimensions of the stems in some of the most prominent mills of California and Nevada.

—	Douglass Mill, Dayton Cañon.	Consolidated Virginia Mill.	Lincoln Mill.	Brunswick Mill.	Electric Mill.	Eureka Mill.
Length . . .	12½ ft.	13 ft.	13 ft.	15 ft.	11⅔ ft.	14 ft.
Diameter . .	2⅞ in.	3⅛ in.	3⅛ in.	3⅛ in.	3 in.	3½ in.
Weight . . .	290 lb.	320 lb.	320 lb.	375 lb.	258 lb.	450 lb.

The tappet, Figs. 77 and 78, is a hollow cast-iron piece, almost cylindrical, weighing from 80 lb. to 125 lb. It is bored out to fit the stem and is 8 in. to 9 in. in diameter, and 8 in. to 12 in. in height. It forms on the stem a projection about 2½ in. wide, upon which the cam catches and lifts the stamp. There are a number

of forms, but the one almost universally used is the gib tappet, Fig. 77. It is cast with a rectangular recess on the inside of the central hole. Into this a piece of wrought iron, *c*, 7 in. to 8 in. long, 2 in. to 2½ in. wide, and ¾ in. thick in the thinnest part,

Fig 77.

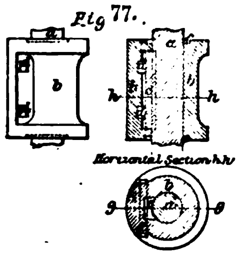
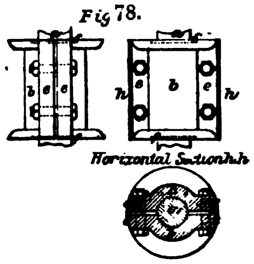


Fig 78.



called a gib, flat on one side, but with the same curvature as the stem on the other, is fitted. Behind this gib are two slots *d*, at right angles to the tappet, which pass entirely through it, and receive the steel keys, which are ⅞ in. by 1 in., and 16 in. long, and which, when driven home, fasten it securely against the stem. It is first bored out in the centre, then moved in the lathe ⅛ in., and rebored. This gives three bearing points, and allows the tappet to move easily on the stem. It is made symmetrical on both ends, so as to admit of being turned round when one end is worn off by the cam. As all parts are liable to wear except a small annular space *f* near the stem, this part is counterbored from ⅜ in. to ½ in. wide, and deep on both ends, so that the edge of the cam will not be worn off by any projecting part of the tappet. Formerly the tappets were screwed on to the stem in order to admit of easily changing the position. This necessitated a screw thread on the stem, and having a key slot cut into it to admit of securely fastening the tappet to its place. This method is now used only in very old mills, and is giving place to the gib tappet, Fig. 77, which is much more easily adjusted, and not likely to get out of order. When its position must be changed it is only necessary to drive out the wedges. At the Douglass Mill, the tappet is made of two pieces of cast iron, Fig. 78, with flanges *e*, so as to be fastened together by bolts, and secured to the stem by tightening the nuts. It has worked for some time very well, and is a very simple construction, but is more likely to get loose

and slip than the gib tappet. Sometimes the lower surfaces, where the cam strikes, are covered with a steel ring, but generally the cast iron is only chilled at that point, and no attempt is made to add steel. The stroke of the cam turns the tappet round; it is generally calculated so as to make one-third of a revolution at every stroke. When set in a mill these tappets run within ¾ in. of each other. The head, boss, or socket, as it is sometimes called, Fig. 79, is a cylinder of tough cast iron, from 8 in. to 10 in. in diameter, and 15 in. to 20 in. high; around each end a band of wrought iron *g*, ½ in. to 1 in. thick, and 1½ in. to 2 in. wide, is shrunk to prevent it from being broken by the stem or shoe. It has in both ends conical openings *a* and *c* to receive the stem and the shoe. The socket for the stem *a* is about 7 in. deep; on the under side a smaller socket *c*, 6 in. to 7 in. deep, is made for the shank of the shoe. From its centre two openings *f* and *e*, at right angles to each other, about 2½ in. by 1½ in., are made, each a little below the bottom of the stem or shoe socket, so as to admit of driving out the stem or the shoe with a steel key when it is necessary to do so. The head lasts a very long time, being rarely ever ruptured. The shoe, Fig. 80, is made either of very tough cast iron chilled on the working face, or of steel. It is composed of two parts, the shank *a*, which fits into the boss, and the shoe proper or butt *b*. They are generally both of the same length, but of different diameters. The butt is 8 in. to 12 in. in diameter, and 5 in. to 6 in. long, and made of cast iron. It is usually chilled to within ¾ in. of the shank. The shank is tapering, and its diameter next the butt is half that of the shoe proper. Its taper is ¾ in. in 6 in. The shoes weigh from 90 lb. to 160 lb. when new, and when worn out from 45 lb. to 60 lb. They are renewed when the butt is worn down to 1 in. in length. They should not be allowed to wear thinner than this, as the boss would be likely to be injured. They are about 1 in. apart in the mortar.

Fig. 79.

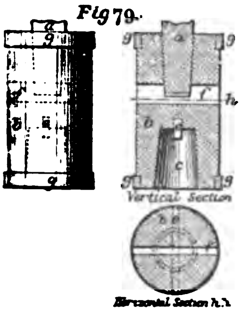
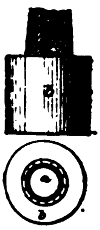


Fig. 80.



The Table below gives the dimensions of different parts of the stamp in several mills.

—	Brunswick.		Douglass.		Consolidated.	
	Height.	Diam.	Height.	Diam.	Height.	Diam.
Shoe . . .	10 in.	9 in.	9 in.	8 in.	7 in.	8 in.
Boss . . .	18 ,,	8 ,,	18 ,,	8 ,,	16 ,,	8 ,,
Stem . . .	15 ft.	$3\frac{1}{8}$,,	$12\frac{1}{2}$ ft.	$2\frac{7}{8}$,,	13 ft.	$3\frac{1}{8}$,,
Tappet . . .	1 ,,	9 ,,	1 ,,	8 ,,	10 ,,	$7\frac{1}{2}$,,

—	Lincoln.		Electric.	
	Height.	Diam.	Height.	Diam.
Shoe	7 in.	$8\frac{1}{2}$ in.	8 in.	$8\frac{1}{8}$ in.
Boss	18 ,,	$8\frac{1}{2}$,,	16 ,,	$8\frac{1}{8}$,,
Stem	13 ft.	$3\frac{1}{8}$,,	$11\frac{3}{4}$ ft.	3 ,,
Tappet	10 in.	$7\frac{3}{4}$,,	$8\frac{1}{2}$ in.	$7\frac{1}{2}$,,
Die	$5\frac{1}{2}$,,	$8\frac{1}{2}$,,	6 ,,	$8\frac{1}{8}$,,

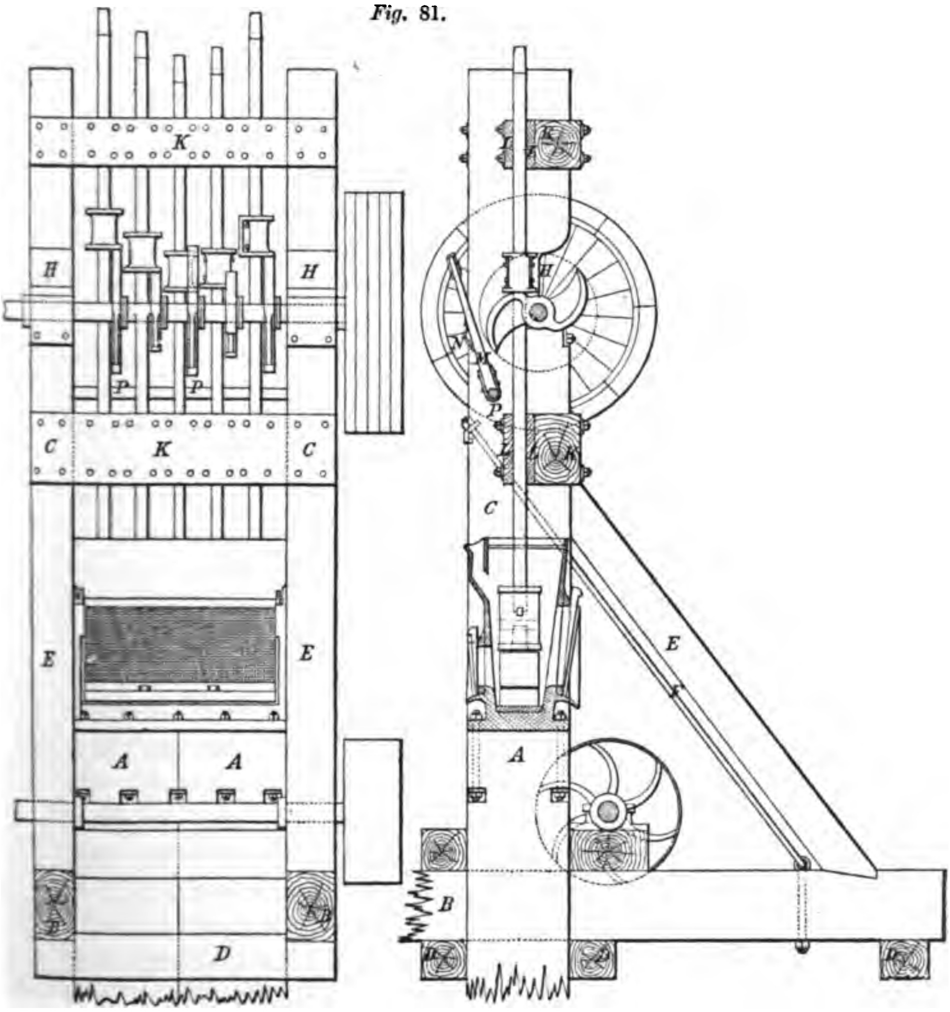
Below is given the weight of different parts of the stamp at different mills.

—	Electric Mill.	Douglass Mill, Dayton Cañon.	Consolidated Virginia Mill.	Keystone Mill.	Lincoln Mill.	Stanford Mill.	Brunswick Mill.	Eureka Mill.
	lb.	lb.	lb.	lb.	lb.	lb.	lb.	lb.
Weight of stem .	258	290	320	350	320	286	375	450
,, tappet .	83	120	95	100	93	114	125	120
,, head .	189	175	175	200	230	280	200	220
,, shoe .	123	115	110	100	119	120	125	160
Total weight of stamp	653	700	700	750	762	800	825	950
Weight of die . .	100	...	...	113	99	...	...	120

The following Table gives the limits of the dimensions of the different parts of the stamp :

—		Length.	Diameter.
		ft. ft.	in. in.
Stem		12 to 15	$2\frac{7}{8}$ to $3\frac{1}{2}$
		in. in.	
Tappet		8 ,, 12	9 ,, 9
Boss		15 ,, 20	8 ,, 10
Shoe	Butt	5 ,, 6	8 ,, 12
	Shank	5 ,, 6	4 ,, 6
Die	Die	3 ,, 4	8 ,, 12
	Footplate	$1\frac{1}{2}$,, 2	10 ,, 14 sq.

Fig. 81.



ARRANGEMENT OF STAMP STEMS.

The stamp stems are guided in boxes L, Fig. 81, bolted to the wooden supports, which run the whole length of the battery, and are fastened to the upright, and serve both as guides and as cross-timbers to strengthen the battery frame to which they are bolted. There are two of these guides K; the upper one is very near the upper end of the battery post, from 1 ft. to 18 in. below the top, and the lower one as low as the raising of the stamp-head will admit, which will be about 1 ft. above the mortar. This makes the two guides 6 ft. or 7 ft. apart. The upper ones are made of timber, 14 in. by 12 in., and 28 ft. long, dressed to 13½ in. by 11½ in.; the lower ones 14 in. by 18 in. and 28 ft. long, dressed to 13½ in. by 17½ in. They are fitted and bolted to the battery posts by 1-in. bolts, and keyed. For the stems oak guide boards L, Fig. 81, are made. They are fitted with wooden keys to hold them half an inch apart; and are secured to the guide timbers by ⅞-in. collar bolts with jam nuts on the front end. The object of the screw is to admit of adjustment when the boxes wear. The stems are fitted as closely as possible to the guides, but when they are light the stroke of the cam bends them, so that the guide boxes are worn as they rise. There is no wear in falling if the stem has elasticity enough not to become bent. With heavy stems the wear is exceedingly light, and amounts to nothing. In some of the best recently constructed mills the guide boards have a square instead of an oval hole cut into them. Into these grooved wooden keys are driven, so that the grain of the wood is parallel to the axis of the stem instead of at right angles to it. With this method the guide boards are never worn and the keys, when necessary, can be replaced in a very few minutes. Iron guide boxes are sometimes fitted to the guides with or without a Babbitt metal lining. At the Eureka Mill the Babbitt metal is used in the iron box; at the Douglass Mill the iron box alone is used; but most mills do not use either the one or the other, but allow the stem to run in the wooden box, which is lubricated with tallow. It has been found that the dirt of the mill becomes attached to the metal, and wears the stem so rapidly that both kinds of metal boxes have been given up. Oak is generally used for these guide boxes when it can be had; but when it cannot, the firmest wood that can be obtained.

Each stamp is provided with a series of fingers or jacks M, Fig. 81, which are made either of wrought iron or hard wood protected with iron. When made of wood, as they generally are, the bottom is fitted with an iron strap, which is the shape of the shaft, and is bolted on with $\frac{1}{2}$-in. bolts. The top is fitted with strap iron $\frac{1}{8}$ in. thick and 3 in. wide, which is fastened by four heavy screws. These are hinged upon an iron shaft P, 3 in. in diameter, and 63 in. long, which is attached to fittings on the inside of the battery post. In the middle of their length is fastened a leather strap or wrought-iron handle N, made of $\frac{1}{2}$-in. round iron, for greater ease in manipulating them. When it is desirable to hang up a whole battery, or a single stamp, the cam is allowed to act until it is at its highest point, the finger is then slipped underneath it. The whole stamp is then supported on the finger, and the cam revolves clear of the tappet. By this method any one stamp, or all the stamps of any battery, may be hung up for repairs, without the necessity of the stoppage of any one of the batteries. This is especially convenient when two batteries have a single shaft, or when any accident has happened which makes it desirable to hang up any part of the battery without stopping the rest. A few inches above the level of the finger shaft, on the feed side of the battery, a platform is arranged to give easy access to the tappets, pulleys, and belting, and to facilitate the repairing of the battery.

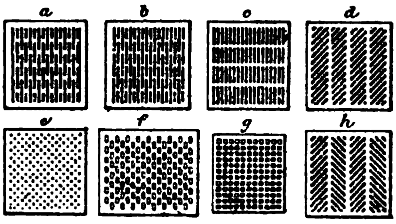
To set up the stamps the stems are placed in the guide boxes, and the tappets placed on them, and fixed in any position. The stem is then hung up on the battery fingers. The shoe is driven into the boss by a sledge and placed on the die, and the stem is dropped into it, or better, the boss is placed on the die, and the stem dropped into it, and this raised and dropped on to the shoe. If the fit is a good one, iron is left against iron; but if not, pieces of canvas about 2 in. wide are dropped into the boss socket, and the stem dropped against them. The outside of the stem of the shoe is covered with strips of wood *a*, Fig. 80, from $\frac{3}{8}$ in. to $\frac{1}{2}$ in. thick and 1 in. wide, which are fastened by a string, and the boss is dropped on it. In wet crushing, these wedges swell and hold the shoe securely. In dry crushing it is apt to get loose, and the stem will not hold, as the boss becomes heated, and the shoes drop

out, so that the wood must be replaced by band iron $\frac{1}{4}$ in. thick. Leather or canvas is sometimes used; these, like the wood, are tied around the conical part with a string and the boss with the stem dropped on to it. It is thus firmly wedged in place. The wood compresses to $\frac{1}{4}$ in. The tappet is now loosened and left hanging on the fingers, and all the stamps are allowed to go down on to the shoe. The whole stamp is now raised, by a block of wood inserted between the shoe and the die, to the height of the fall. The tappets are then fastened, the blocks withdrawn, and the battery finds itself hung on the fingers.

The screens for dry crushing are made of brass wire cloth, with from 30 to 100 holes to the linear inch, or from 900 to 10,000 to the square inch. They are set in battery frames, so as to be easily replaced. For wet crushing Russia sheet iron or sheet steel is used, which weighs about 1 lb. to the square foot, and is about $\frac{1}{32}$ in. thick. It must have a smooth glossy surface, be very soft and tough, and free from rust or flaws of any kind. It is usually tested by hammering up some concave object with it. If it stands this test it is safe to use it. The holes may be round, or they may be long slots arranged parallel or inclined to each other. The size is regulated by the numbers of sewing machine needles from zero to 10. No. 8 is $\frac{1}{64}$ in. and No. 5 about $\frac{1}{16}$ in. In round holes the diameter is given in these numbers; in slots the width of the slot. These holes for silver crushing usually vary from $\frac{1}{40}$ in. to $\frac{1}{24}$ in. in diameter. The usual slot for a gold mill is $\frac{3}{8}$ in. long, and of the width of the diameter of a No. 6 sewing needle. There are two kinds of these screens, called the clean and the indented. The latter is rough on the side of the inner edges, which is thus slightly smaller than the other. The rough side is always on the inside of the mortar. The object of this is, that when the inside wears so that the slot becomes wider, they may be beaten together with a mallet, and the screen be made to last longer, but the screens are apt to crack and break before this is done. The life of a screen will thus be that of the iron of which it is made, and there consequently does not seem to be much advantage of the clean over the indented slot. There is the greatest possible variety in the arrangement of the slots. Some of these are shown in Fig. 82, *a* to *h*; they are sometimes

made vertical and parallel to each other, *c*, sometimes with the end of one opposite the middle or end of the next, *a* and *b*, sometimes slanting, *d*, sometimes with two at right angles to each other, the angle being placed at 45 deg. with the horizontal *h*, and sometimes vertical as *c*, Fig. 82. There does not seem to

Fig. 82.



STAMP MILL SCREENS.

be any special advantage in the methods $d \times h$. The holes also are rectangular, *a* to *c*, round, *e*, oval, *f*, or D-shaped, *g*, the advocates of each claiming some peculiar advantage for it.

Formerly in all dry crushing mills screens from No. 60 to No. 80 were used under the idea that the amalgamation would be improved the finer the ore was crushed. It has been proved however, by experiments made at the Ontario Mill,* that this is not the case, but that it is rather an advantage to crush coarse. Exactly what the size will be depends upon the ore, and must be ascertained in every case.

The difficulty in the battery is not to crush the rock, but to get rid of it after it is crushed, as the crushing power is greater than that of the discharge; hence the great number and variety of the screens. It would seem that the greatest delivery would be from some kind of a slot which is either vertical or horizontal, but every variety has its advocates, and the matter has not as yet seemed of sufficient importance for any one to make conclusive experiments on it. There is, therefore, a great diversity of opinion as to whether holes, slots, or wire cloth are the best. Slots seem to be preferred as better suited to the discharge than meshes, but it may fairly be questioned whether what is gained in rapidity of discharge through the slot, is not more than compensated for by the loss of discharge area from the wire cloth.

* "Mining and Engineering Journal," vol. xxxv., p. 348.

The screens last in dry crushing from three to four weeks. The middle screen lasts longest. The dies when new come to within about 1 in. of the lowest portion of the discharge. It is desirable to make this interval, which is called the height of issue, as small as the screen will bear. It was found at the Metacom Mill that after the dies had worn away 1½ in., the introduction of new ones raised the capacity of the whole battery nearly two tons. A five-stamp wet battery requires on an average thirteen sets of screens a year. A set consists of five sheets of 1 to 1½ square feet.

These screens are securely fastened in iron frames which fit into slots prepared for them in the mortar. Formerly they were placed vertically, but this is now rarely done. It was found that the discharge was facilitated by placing them at an angle which is generally determined in each particular case, and is not far from 10 deg., which angle has been found to be the best for dry crushing. The screen should be as high as possible.

When the throat of the battery is low, or when it is open, the effect of the fall of the stamp will be to throw the pulp vertically as well as horizontally, and some of it may be thrown back on to the feeding floor. This indicates a fact that is often overlooked in the construction of mortars; that since the impulse given to the pulp by the stamp is radial in all directions, with a decided upward movement at the same time, the greater the surface of discharge, the greater the stamp duty will be. With a discharge 18 in. high the fine pulp comes mostly through the upper 6 in. of the screen, and in batteries lower than this will fall back into the mortar, and remain there until it is discharged below. In order to get the maximum discharge surface, it is now generally the practice to use a double discharge, having screens both in front and behind, the feeding being done over the rear screen. To secure the maximum discharge, round mortars, in which the screens occupy the entire circumference, and containing only a single stamp, have been proposed, but while the discharge in such a mortar is a maximum, the expense for construction is also a maximum, and they have not come into favour. It has been proposed to place screens on the ends of the mortar as well as the sides, and these have found some advocates, but are not in general

use. Straight screens must as a general thing be used, because it is exceedingly difficult to fasten a curved screen into the mortar in such a way that it will not become loose and break. In dry crushing a rear screen is not generally used, as it breaks readily from coarse ore being thrown against it. Single and double mortars with end screens, however, have their advocates.

In wet crushing it seems generally best to have front and rear screens. As a rule the stamps will crush faster than the material can be discharged from the mortar. It was found, after a series of experiments made at the Metacom Mill, that the pulp put directly back into the mortar took about as long to go through the screens as fresh rock, and the same has been found to be true in the Ball stamp. This is more especially true with slow running and dry crushing, as the fine dust has a constant opportunity to fall back instead of passing out. An exhaust fan in dry crushing is used to assist the discharge and at the same time to draw the dust away from the wearing parts of the mill. The material discharged through the screens is, when the ore is rebellious, carried by conveyers at once to the supply bins of the roasting furnaces, so that there is little or no dust about the mill.

The screens in wet crushing are keyed into the mortar with iron keys, so that they may be easily removed. In dry crushing they are screwed on with bolts; the screws are completely covered by a box, to which various devices for driving out and carrying away the pulp, such as endless belts, &c., are attached. In wet-crushing gold mills, the splash-box is fastened to it by means of bolts with a blanket packing. It generally has three spouts or openings to direct the discharge on to blankets or amalgamated plates in different directions. In wet-crushing silver mills there is no splash-box, properly speaking; the discharge is protected by a wooden box, and flows into a trough, which carries it to the settling vats.

In Colorado the mills supply their own water. In Nevada, in the vicinity of Virginia City, the water is purchased from the Virginia and Gold Hill Water Company. In other districts the mills supply their own water, the Eureka Mill using the whole supply of the Carson River in summer.

In Western California, near the hydraulic mining region, the ditch companies supply the water; remote from this district, the mills supply their own. The water consumed in Nevada and California is 200 to 300 cubic feet per ton of rock stamped, or from $\frac{1}{3}$ to $\frac{1}{2}$ cubic foot per stamp per minute. This includes, however, all the water used in the mill, including the pans, which water does not pass through the batteries, and which amounts to perhaps from $\frac{1}{8}$ to $\frac{1}{12}$ cubic foot per stamp per minute, leaving about $\frac{1}{4}$ cubic foot, as used in the batteries. In Colorado the amount is 28 cubic feet per ton of rich ore, and 33 cubic feet of poor ore. The cubic foot of ore will average from 108 lb. to 125 lb. This will be $\frac{1}{4}$ cubic foot per stamp per minute, or about the same as that used in Nevada and California. In the Brunswick Mill the water is supplied by a 3-in. pipe, with a head of 2 ft. The water when purchased is measured by the miner's inch, which is the quantity which will pass through an orifice in the measuring box 1 in. square, under a head of 6 in. The aperture is generally 2 in. wide, and the length is determined by the quantity desired. The water does not always stand 6 in. in the measuring box, but is sometimes 9 in. to 10 in., so that the miner's inch is not fixed. An inch of water is by common consent, according to Ross Brown, accepted as 4032 cubic inches, or 145.86 lb. per minute, 3360 cubic feet, or 10,656 gallons in twenty-four hours, and 1,226,400 cubic feet, or 30,410 tuns of 40 cubic feet each.* In the year 1867 an effort was made to establish a miner's inch by law, and it was proposed that it should consist of $2\frac{1}{3}$ cubic feet of water, or 7.4054 gallons of water, or 145.86 lb. of water per minute, passing through a given orifice. The Legislature declined to pass the law, on the ground that the companies selling water had the right to agree upon their measurement and prices of water, and that companies had already been organised on a different plan.

The water is supplied to the stamps by horizontal pipes passing just above the feed slot of the mortar, with openings made opposite to each stamp, which can be closed if necessary; or sometimes the pipe is carried higher up, and vertical pipes are conducted from the horizontal ones with valves to shut the water off.

* See Chapter on Hydraulic Mining, vol. ii.

The conduit pipe is usually a gas pipe about 3 in. in diameter, but its size depends upon the supply which is necessary. There is also a supply pipe of about half the size in front, to help carry off the pulp. Arrangements are made in all the mills to heat this battery water by means of steam in the winter, when steam power is used, or in some other way when it is not.

The water pipes of the sluice in place, the battery is ready to run. When the stamps are new the first material run through them is barren rock, to scour the mortar and get the batteries, shaft, and boxes in working order. During this time the sluices are turned into the mortar block trench until it is entirely filled. The jar of the stamp settles the sand, the water helping to pack it very tight. The excess of water is allowed to run over the top of the trench, or is pumped out, as the case may be; by the time that this is done the battery is usually ready to run.

The rock, if the mine belongs to the mill, is brought from the mine and dumped on a screen to separate all the rock ready to go to the stamps, which passes at once into the feed-box; what does not pass the screen is delivered directly in front of the rock breaker, which is almost universally Blake's.* This is done both to avoid giving extra work to the crusher, and to separate pieces of iron, splinters of wood, and other material which might go into the battery and clog the screens. In all custom mills, ore is delivered in wagons, sampled, and put into separate bins for treatment. At the Eureka Mill the ore goes over two screens made of 1½ in. round iron, the bars set 2 in. apart. All that will not pass through goes to the Blake's crusher, the rest to the stamps. One man at the crusher and one at the screens per shift handle 185 tons in twenty-four hours. The man at the screens hauls the cars and dumps them.

It is generally desirable to crush fine, since the particles of gold and silver must be separated and amalgamated mostly by gravity. It is also desirable to set the jaws of the crusher near together, so as to decrease the quantity of work which the stamp has to do. A Blake's crusher, making 170 strokes of ¾ in. per minute,

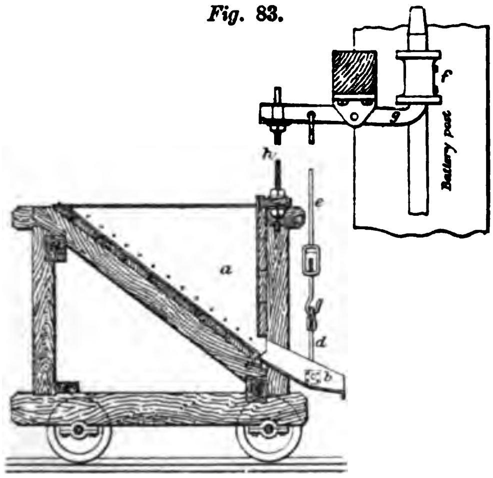
* This crusher was first introduced in Colorado in 1864 and has since then been almost universally used. It both economises the work and increases the possible output of the mills.

will prepare 72 tons of rock, and will run, provided it is properly fed, a 30-stamp mill for twenty-four hours. The crusher is not usually run at night unless the supply of ore is short.

As the stamps at the end of a mortar are liable to do less work than the others, it would seem that the greater the number there were in one mortar, the greater the average work would be. Experience, however, has shown that six are too many, on account of the necessity of increasing the strength of the cam-shaft, and preventing its springing, and that four are too few, as they do not do a sufficient amount of work; so that most of the modern batteries are constructed with five stamps running in one mortar. The order of drop is different in dry and wet crushing, and different in different mills of each kind. There are evidently two extremes, which are, dropping all the stamps at once, which would probably break the screens, strain the engines, and would in a very short time rack the stamp frame to pieces; and dropping them in serial order 1, 2, 3, 4, 5, which would drive the ore to one end of the mortar, and make the stamps there do nearly all the work. The orders 3, 4, 2, 1, 5—2, 4, 5, 3, 1—3, 5, 1, 4, 2—3, 4, 5, 2, 1, are generally used in the Nevada and California mills; the last one particularly makes a wave backward and forward, and keeps the mortar very evenly filled. On this account all of these orders are very much used. In other mills, however, the order of dropping the end stamps first is very much preferred, which would be 1, 5, 2, 4, 3, which is the drop used at the Eureka Mill, and 1, 5, 4, 2, 3, which is also extensively used, and seems to be equally advantageous with the other. The object to be secured in all cases is to have an equal amount of ore under each stamp, so that each one shall do the same amount of work, and produce an equal discharge from each screen. A maximum discharge seems to be secured by allowing the middle stamp to drop first. The maximum amount of work appears to be done by allowing the end stamps to drop first; hence there are at least six or seven ways, all of which have their advocates, which seem to be, so far as the general working and the wear and tear of the mill are concerned, about equally good. In all crushing the objects to be secured are, first, the equal distribution of the ore between the shoes and dies so as to give an equal power to each stamp; and, second,

a maximum discharge of pulp from the screens. This maximum discharge seems best to be obtained by dropping No. 3 first, as it is found that the central stamps receive and deposit nearly all the ore; but a great deal depends upon the judgment of the feeder. On this account the automatic self-feeders which are being introduced every year more and more, meet with determined resistance, because they have, up to this time, been arranged to feed exactly the same quantity, and make no difference in the work for different parts of the mortar under different conditions. The batteries are usually fed by hand. The ore is received into a large bin or pocket, the floor of which is made in such a way that the ore will run easily toward the stamp. The feeding slot of the mortar is always placed behind, and a space of 6 ft. to 12 ft. left behind the stamps for the feeders. Their duty consists in keeping the shoes off the dies, and having a certain amount of ore, generally about 2 in., always in the battery at the end of every stroke, so that the fall of the stamp will always be the same, and not of variable heights. A good feeder knows by the feel and sound of the stroke whether the stamp is being properly fed or not; but as intelligence is a thing not to be depended upon, the effort is being made now, with more or less success, to replace

Fig. 83.

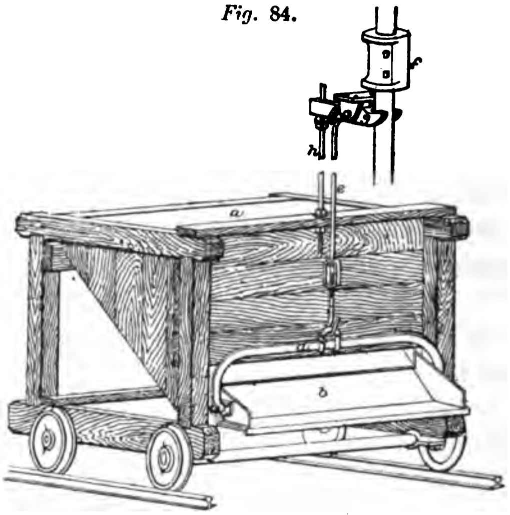


STANFORD'S AUTOMATIC FEEDING APPARATUS.

hand feeding by automatic feeding. Automatic feeders have been used with more or less success. They are of the same

general type, and consist of a pocket to receive the ore from the shoots, the front of which is movable, or the bottom is provided with an endless belt, which is forced forward when the ore gets low. Some of these are applicable both to dry and wet ore, and some to dry ore only. One of those which is extensively used, Figs. 83 and 84, is that invented by C. P. Stanford (who is said to

Fig. 84.



STANFORD'S AUTOMATIC FEEDING APPARATUS.

be also the inventor of the California stamp mill), which consists of a hopper *a* with an adjustable spout *b* at its end. This spout is swung on trunnions *c*, and is attached to a crossbar *d*, to which an upright adjustable rod *e* is attached. On the top of the stamp stem a feeding tappet *f* is keyed. A lever *g*, rotating on pivots, is attached to the rod *e* so as to be struck by the cam tappet. This lever is forked, so that it spans the stem. When the battery is full of ore the tappet does not come down far enough to strike the end of the feeding rod; when the ore gets low it does strike it, and the result is an oscillating motion of the spout *b* on its trunnions, which throws the ore forward. The quantity of ore under the stamp will, therefore, always be the same, and it is said that the work of the battery is increased one-fourth. The whole machine is placed on wheels, so that it can readily be moved to one side. It is very simple, and is not in the way. The height of the ore in the battery can always be regulated by this feeder, by simply adjusting the tappet so that

it will always strike at the same position. It is not likely to get out of order, and it is claimed that it decreases materially the wear of the shoes and dies, since it is impossible that they should strike the one upon the other.

Another machine has a drum divided on its circumference into semicircular chambers. The shake produced by the stroke of the tappet causes this drum to revolve, and in doing so each chamber discharges the ore contained in it into the mortar. Another class of machine, invented by Cochran, has a cam held taut by a spring, which, when struck, causes the bottom of the feeder, which is an endless leather belt, to advance, throwing the ore into the battery. This can hardly be used for wet ore, though it works very satisfactorily for dry. All of these feeders are constructed on the plan of shaking the ore into the stamp hopper. This plan answers very well for dry ores, but sometimes does not give satisfaction when they are wet. For this class of ores, feeders which drop the ore at intervals seem to answer an excellent purpose. Intelligent management has, however, almost more to do with the regular feed than the machine. All these feeders have been in use in the Keystone Mill, and gave perfect satisfaction. They all seem to answer the purpose, though Stanford's seems to have the preference on account of its extreme simplicity and ease of repair. The Eclipse and Hendy's challenge feeders are also much used. When the feeding is done by hand, as at the Eureka Mill, one man is sufficient for twenty stamps on an eight-hour shift.

The Keystone Mill has 40 stamps in 8 batteries, two of which are served by Cochran's and one by Stanford's feeder. These feeders and one man feed the whole battery. At the Brunswick Mill, which has 56 stamps, one man is required at the crusher, one to draw the ore, and there are three battery feeders at eight-hour shifts.

It was formerly a vexed question whether it was best to have light or heavy stamps, and whether a slow or quick motion was most effective. A series of experiments were made some years ago at the Metacom Mill, the results of which have been a change in the weight and construction of the stamps, which are now made much heavier than they formerly were, and run at a

much higher velocity. The horse-power used to run a battery varies directly with the weight of the stamp, which varies between 700 lb. and 950 lb. Hence it has been necessary to ascertain for every ore, whether heavy blows do as much work proportionately as very quick light ones. In discussing the question it should be taken into the account that in dry crushing the blow of the stamp both pulverizes the ore and produces a compression of the air, which, together with the blow, forces the pulp out of the mortar, and in dry crushing this is the only force used to free the mortar. The slower the stamp runs the less constant this air compression is, and the more likely the pulp is to fall back into the shoe, and hence to get a maximum discharge, the stamp must be run rapidly; in wet crushing the force of the water helps to force the pulp through the screens. It was found, by a series of experiments in dry crushing at the Metacom Mill, that with 60 drops per minute, 4½ tons, or not quite one ton per stamp, could be crushed and discharged from the mortar in twenty-four hours, 90 drops discharged 10 tons, and 102 drops 15½ tons, or a little over 3 tons per stamp; the increase of speed thus increasing the yield of the battery 244 per cent. To this must be added the economy in wages and interest on the capital, &c., owing to the greater efficiency of the stamps. The difference is not so great in wet crushing. Here the stamp yields about 1¼ tons per horse-power developed by the stamp. With the velocity of 105 strokes per minute, which were made during these experiments at the Metacom Mill, nearly all the old mills and many of the modern ones would have been racked to pieces, owing to the fact of their foundations not having been sufficiently well built to withstand such rapid strokes. With this velocity the rebound of the stamp stem was about 1¼ in. It necessarily follows that with such a high velocity the cams were single. Double cams at such a velocity would not have allowed the stamps to fall at all. One of the great advantages of the high velocities is that the feeder is kept constantly busy; he must give his undivided attention to his work, in order to keep the shoes off the dies, while with low velocities he has frequent intervals of repose, and times when he will be likely to think of something else besides his work.

The question of both weight and speed must be determined by experience for each ore. The tendency formerly was to run light stamps with a high drop and low speed, but the general result of experience so far has been in favour of a maximum weight and speed, with a low drop; for there has always been a gain in the ore crushed more than proportionate to the increase of speed.

The following Tables, the details of which were supplied by Booth and Co., of San Francisco, show conclusively that heavy stamps, with a low drop and high speed, will do more work than light stamps with a high drop and low speed:

Dry Crushing Silver Mills.

—	Stanford Mill, at White Pine.	Raymond and Ely at Pioche.	International Mill, at White Pine.	Lexington, Montana.
Number of mortars	6	6	6	10
Discharge of mortars	Double	Double	Double	Single
Number of stamps to each mortar	5	5	5	5
Total number of stamps	30	30	30	50
Weight of a stamp in pounds	750	750	750	850
Height of drop in inches	8	8	7½	7 to 8
Number of drops per minute	95	95	93	94
Screens made of brass wire				
Trade number of the screens	50	50	50	30
Tons of rock crushed in 24 hours	52	48	33	60
Tons crushed per stamp in 24 hours	1.73	1.6	1.1	1.2
Quality of the rock	Hard	Easy	Soft	Hard
Formation	Limestone	Quartz	Limestone	
Fineness of the bullion	.998	.775	.990	.980

It is evident that a heavy stamp requires more horse-power, but it also does more work. There is evidently a limit for the velocity and the weight; for the cam-shaft may be made to run so quick as to keep the stamp constantly in the air, and the weight may be so great as not to be economical in the use of power so far as the ore is concerned, and also to rack the battery. It will be found that the stamps, as a general thing, crush faster than they can discharge, for if the crushed ore is put immediately back into the mortar it will take about as long to discharge it, as

WET CRUSHING SILVER MILLS.

—	Meadow Valley Mill at Pioche.	International Mill, at White Pine.	Eureka Mill at Carson River, near Virginia City.
Number of mortars	6	6	12
Discharge of mortars	Double	Double	Single
Number of stamps to each mortar	5	5	5
Total number of stamps	30	30	60
Weight of a stamp in pounds	750	750	950
Height of drop in inches	9	7½	9
Number of drops per minute	85	87	90
Screens made of Russia iron	Punched	Punched	Punched
Trade number of the screens	6	6	4
Tons of rock crushed in 24 hours	67	47	159
Tons crushed per stamp per 24 hours	2.17	1.57	2.65
Quality of the rock	Tough	Soft	Easy
Formation	Quartz	Limestone	Quartz
Fineness of bullion	.550	.990	.980

WET CRUSHING GOLD MILLS.

—	Keystone Consolidated, Amador County, California.	Hunter's Valley Mill, Mariposa County, California.	St. Lawrence, Newcastle, Placer County, California.	Ontario, Utah.	Lexington, Montana
Number of mortars	8	6	1	6	10
Discharge of mortars	Single	Single		Single	Single
Number of stamps to each mortar	5	4 with 4 2 „ 6	6	5	5
Total number of stamps	40	28	6	40	50
Weight of a stamp in pounds	750	650	650	850	850
Height of drop in inches	8½	11	10	8½	
Number of drops per minute	85	70	90	92	94
Screens made of Russia iron	Slotted	Punched	Punched	Punched	
Trade number of the screens	5	6	5	30	
Tons of rock crushed per 24 hours	90	50	17	60	
Tons crushed per stamp per 24 hours.	2.25	4 stamp 1.75 6 „ 1.83	2.85	1.5	
Quality of the rock	Medium	Easy	Brittle	Hard	
Formation	Quartz	Quartz	Quartz		
Fineness of the bullion	.840			.980	

fresh ore. Working slow would aggravate this difficulty, because the crushed material would have plenty of time to fall by gravity again beneath the shoe. Working fast, by keeping the pulp constantly in motion, tends to prevent any of it accumulating in the mortar, and thus increases the amount of work in a given time. In dry crushing an exhaust fan is sometimes used to draw the crushed ore out of the mortar.

The number of drops is usually between 70 and 100. The usual fall is from 7 in. to 10 in., varying, of course, with the velocity of the stamp. The greater the number of blows the less the height, and hence with a very high velocity single-toed cams must be used. At the Keystone Mill, with a stamp of 750 lb., with double-toed cams, the drops are 75 to 80; at the Eureka Mill, with a 950 lb. stamp, with double-toed cams, they make 80 drops of 9 in.; at the Idaho, also with double-toed cams, and with a stamp of 950 lb., they make 60 drops of 10 in.; and at the Metacom Mill, with a stamp of 900 lb., and single-toed cams, they make 90 drops of 10 in.

The quantity stamped will depend upon whether the mill is for wet or dry crushing, upon the character of the ore, and upon the facilities of discharge. It will generally be from one to four tons per stamp in twenty-four hours, depending upon the character of the rock and the weight and velocity of the stamp. In California it is usually from one to one and a quarter tons in twenty-four hours for every horse-power developed by the stamp. In dry crushing it is about 0.45 of a ton. In Grass Valley a battery of 20 stamps, each weighing 850 lb., with 61 drops of 10 in. a minute, crushed 40 tons of quartz in twenty-four hours, without a rock breaker; while a battery of 20 stamps, weighing 700 lb. each, with 68 drops of 10 in. a minute, crushed 32 tons of the same rock; the same screens being used, and the same conditions observed in both trials. This is a very good example of the effect of weight on the work produced. At the California Mill in Virginia City, which has 80 stamps, they make 90 to 100 drops of 7 in. to 8 in. per minute. The ore is soft, and 360 tons are crushed in twenty-four hours, making 4½ tons per stamp per day. The Virginia Consolidated Mill, with a moderately soft rock, averages about 2 tons per stamp in twenty-four

hours. Some mills give higher results and some not so high. The Eureka Mill, on the Carson River, has 60 stamps and crushes 2½ to 3 tons per stamp. The Brunswick Mill, near Carson City, has 56 stamps and crushes 160 tons, or nearly 3 per stamp. The Keystone Gold Mill, in Amador County, California, has 40 stamps, and crushes from 75 to 80 tons per day, or not quite 2 tons per stamp. It formerly did 2½ tons, but on account of a change in the process which necessitated a finer pulp, less work could be done.

The wear of different parts of the stamp is very variable. The stems rarely ever wear out, those at the Keystone Mine, in Amador County, California, have been in constant use for three years. They are more likely to wear in metal than in wooden guides, their wear depending on the amount of dust which remains attached to or incrusted in the guides. They break off at the ends more frequently when the stamp is fed by hand than by a self-feeder; and generally when the shoe and die are carelessly allowed to come together. This should not happen in a small mill oftener than once in two or three years when the stems are new: when they are old and the men careless it may happen once in three or four months. In very large mills a stem will be broken on an average once in two weeks. The stem is then turned over, and when both ends are broken new pieces are welded on, and they are used in this way an indefinite number of times.

The tappets have no very considerable wear. They occasionally break in setting up the battery when the wedges are driven in too hard. They should last three to four years at least.

The heads, when they are strengthened with iron rings, do not wear out; when they are not thus strengthened they occasionally split. The only wear they are likely to have is in the sockets for the shoe and stem, but even when they become a little enlarged they can still be used by putting in either iron or wooden strips to serve as wedges.

The time that the shoes and dies will last, depends upon the quality of the iron with which they are made, and on the hardness of the rock they have to treat, the charge that is put into the battery, and the speed with which it is run. It is

thought that heavy charges and high speed produce less wear than small charges and low speed, for the reason that at a high velocity the feeder is kept busy all the time, and does not wait before charging until he ascertains by the noise and the jar that the shoe is pounding on the die. With soft rock they have been known to last for five months; with rock of average hardness they are generally worn out in from four to six weeks. It will generally be found that the wear of the shoes will be from 1 lb. to 1½ lb. per ton of rock stamped, depending upon the hardness of the rock. At the Douglass Mill, in Dayton Cañon, they last 38 days. At the Keystone and Eureka mills, near Virginia City, they last 50 to 52 days. The die is then worn down to the foot-plate, and the shoe is about an inch thick. On pure quartz they will not last longer than from 20 to 30 days. Steel shoes are coming into use. They wear about the same time as the best cast-iron shoes, and are therefore preferred to any other in many places. There is always an uncertainty about the quality of the cast-iron shoes, especially in remote districts where the foundries use all and any cast iron they can get. As the steel has to be sent to the mill, and cannot be cast in the vicinity, it is of more even quality, which seems to be its only advantage over cast iron. A good cast-iron shoe will do as much work as a steel one.

The wear of the die is only about half as much as that of the shoe, and will generally amount to from three-fifths of a pound to a pound per ton of rock. It lasts from six to seven weeks. When worn to the foot-plate it is taken out and replaced. The cost of the wear and tear of a 30-stamp battery taking the average of some of the best mills in Utah, Nevada, and Montana, has been found to be for twenty-four hours' use :*

Cost of all parts subjected to wear and breakage screens, supplies interests, &c.	$11.50
Labour for repairs...	5.50
Total	$17.00

As the winters in the high elevations where these mills are usually built are very severe, it is necessary to cover the stamps

* "Production of Gold and Silver in the United States for 1883," page 742. Washington, 1884.

with a building which must be very strong, not only to resist the jar of the stamp, but also the winter winds. These houses are of unusual solidity.

The cost of a large mill, including the house, engines, &c., will be not far from $700 to $1000 per stamp, depending on their number, more for fewer and less for a larger number of batteries.

The power used to run the battery is usually a high-pressure steam engine; the fuel being wood, which was formerly the only fuel of these regions. The stamps and all the other machinery connected with the mills are driven by belts. The amount of fuel required to run a battery will depend upon the weight of the stamp. It will usually be found to be .22 to .25 of a cord of wood per stamp. Sometimes, as in the case of the Brunswick, Eureka, and other mills, in Nevada, the stamps are driven by water power, but it not unfrequently happens, as in the case of the Brunswick Mill, that the water is either too high in the river in the spring, in which case it backs up against the wheel and prevents its working, or too low in the summer, when there is not sufficient power to run it. The mill men have not yet learned that water, being constantly subject to maxima and minima, is an uncertain power to depend upon. The Eureka Mill, near Dayton, Nevada, is run by a Leffele's turbine wheel of 409 nominal horse-power. It is 52 in. in diameter, makes 160 revolutions, and consumes 6777 cubic feet of water per minute. The flume which supplies it is 14 ft. wide, 5 ft. high, with 36½ ft. fall to the mile. It is three-quarters of a mile long, and takes the entire water of the Carson River during the summer. The best economy in all stamp mills is to have such character of engines that they will run from two to three months without stopping. The delays caused by shutting down the mills, even for a few hours at a time, represent such a loss of output that any economy gained in the purchase of an engine that will not do such work is entirely exhausted by the diminished output.

Accidents are generally the result of carelessness, and are caused by the loosening of the nuts and bolts, the breaking of the cam, allowing the shoe to get so thin as to break when the boss comes down upon the broken pieces, or breaking the stem. Most

of them may be caused by the striking of the shoes upon the dies from careless feeding, or by allowing the shoe to wear too thin. If the mortar is filled too high, since the boss is only held in the stem by friction, it may be that the stem will be drawn out, and at the next revolution it will come down upon the boss and either break it or batter or break the stem. Cams are generally broken by carelessness of the engineer in running the cams in the opposite direction, or by carelessly drawing out the fingers so as to let the stamp fall on to the cam ; or they may be worn so thin from the neglect of lubrication as to break by their shock against the tappet. There are besides this a few unavoidable accidents which are generally confined to the stems or shoes, or the wearing out of the screens. The most frequent accident is the breaking of the stem. When it has been broken only once it is simply necessary to take it out and turn it over, both ends being tapered for this purpose. When it has been broken at both ends it must be taken out to be repaired, and a new stem put in in its place to save time. The stem may be repaired a number of times provided that after each welding the whole stem is annealed. When this is not done they frequently part at or near the weld after a short usage. A little attention to this matter of fatigue through want of annealing would diminish the repairs to the stems so as to make them almost nothing. To put in a new stem the whole battery is first hung up on the fingers, the boss and shoe are taken out from the mortar, the stem is then fastened to a rope or chain which is attached to a pulley, the tappet is then loosened and the stem removed. In all the best recently constructed batteries, a chain pulley running on a small railway is provided over all the stamps. The stem is turned over, or a new one put in and held by the pulley, it being slipped through the tappet and guide boxes while it is still held by the rope. The boss and shoe are placed on the die and three or four strips of wood, 1½ in. wide, are put into the stem socket exactly as if the battery were new. The stem is then let down to within about 3 in. of the boss, and the tappet keyed on in this position, and the stem with the tappet hung up upon the finger. While the whole of the rest of the battery remains hung up, a strip of wood 2½ ft. long and 2 in. wide, covered with leather, is placed

between the cam and the tappet and held in one hand, while with the other the workman seizes the handle of the finger and pulls it out from under the tappet. The stem falls into the socket of the boss. Keeping the stick in its place, the workman allows this single stem to work until it is quite fast in the boss; he then hangs it up again, and, going below, raises the shoe to its proper level, supporting it there, and then sets the tappet in its proper place. This is a delicate operation and requires considerable skill on the part of the workman, for, for a certain time, the stem is held up by the pulley alone, and just the right time must be taken to set the stick under the cam. If the stick is not held exactly in the proper position it will be thrown out of the hand of the workman, and may result in breaking the stem. The whole operation of changing a stem takes about half an hour.

In former times in the wet crushing mills the ore had to be dried before it could be roasted, and this was done, as in the methods described in the various mills where wet crushing is done, by means of the waste heat from the furnace running through flues covered with an iron plate which formed the floor of the works. This is not only bad for the health of the workmen, who must constantly walk on the hot iron plates, but is uncertain with regard to time. It has been abandoned in most of the mills, and the principle of revolving driers, as described in the chapter on leaching, or of shelf driers,* which were introduced into the Lexington Mill in Montana in 1882, is coming rapidly into use, the principle adopted being to make them as far as possible automatic.

The size of these mills is fully illustrated in the chapter on pan amalgamation, where the plans of some of the largest mills ever constructed in the United States have been given. The dimensions of the Lexington Mill, which is one of the largest of those recently constructed, are given below. The entire structure measures 326 ft. greatest length, 138 ft. greatest depth, covering over 29,000 square feet.† It is distributed as follows:

* Stetefeldt Shelf Drying Kiln, Trans. Am. Inst. Min. Eng., vol. xii., p. 95.

† "Engineering and Mining Journal," vol. xxxiv., p. 255.

	ft.		ft.
Dry kiln building	120	by	27
Battery ,, ...	120	,,	32½
Stetefeldt furnaces, each	103	,,	47
,, ,, firemen's floor, each ...	33	,,	13
Pan room	163	,,	65
Agitator room	24	,,	22
Engine and boiler room	70	,,	43
Bullion furnace	35	,,	30

The fall from the charging floor of the dry kilns to the boiler room is 61 ft.

The battery dust chambers of the Lexington Mill are constructed as follows: in front of the battery, and suspended overhead, is a wooden box or chamber 110 ft. long, 5 ft. wide, and 6 ft. high. The bottom of this chamber is formed by 22 sheet-iron hoppers, 5 ft. square, which end each one in a canvas hose 4 in. in diameter. Each battery housing for five stamps is connected with the chamber by a stove-pipe 8 in. in diameter, standing at an angle of 75 deg. Across the chamber are partitions which force the current of air, created by a Sturtevant suction fan, up and down, thus facilitating the settling of the dust. The draught given to each battery is regulated by a damper. The dust collected in the hoppers can be taken out at any time through the canvas hose attached to each hopper. It is taken to the dust hopper, mentioned above, and is at pleasure gradually mixed with the pulp coming from the battery. This system works very well and is being introduced. With a strong draught, the dust which settles in the dust chambers amounts to about 1 per cent. of the ore going through the battery.

The Ball Stamp.

Another stamp, known as the Ball stamp, although but little used among the silver and gold mills, on account of its great capacity, merits attention as showing the direction in which the improvement in stamp mills at one time tended. The conditions of the copper and the precious metal mines are not exactly the same, but when a very large output is required this stamp might

be used to crush gold and silver ores. It is now used for crushing copper rock in all the best ordered and recently-constructed mills on Lake Superior. As shown in perspective Fig. 85, and sections, Figs. 86 and 87, it is an overhead steam stamp of great power, having, in comparison with other stamps, an enormous daily capacity. It is generally constructed in the East and sent out from there. It requires very careful construction of the foundations, as the stamp itself is very heavy, and the blow is therefore a very powerful one. To construct the foundations, a pit 12 ft. deep is dug, round which a wall is built 3 ft. to 4 ft. thick. The pit is 14 ft. by 16 ft. inside of the wall. On the bottom of the pit, an anchor-piece weighing about 3 tons is placed. The pit is then filled to the top with timbers 14 in. square, laid across each other in alternate rows. Bolts pass through them to the anchor. The joints of all the timbers are filled full with cement and rammed tight, each tier being cemented as it is put in. A space of about 1 ft. all around is left between the timber and walls, which is not filled until all the work is in. The long bolts passing through the timbers are occasionally broken. To repair them a long lag-screw is put in, which is simply a bolt with a wood screw filed on it. A new hole is bored out and the wood bolt screwed into the foundation timbers.

The timbers A A, Figs. 86 and 87, are 12 in. by 14 in. in size, and are placed on the top of the foundation, spaces having been left between them sufficient for a number of bolts to pass. Upon the top of this frame two cast-iron sills B, with very broad bases, are placed, which are secured to their position by bolts which pass through to the bottom of the foundation timbers and anchor plates. These secure the stamp to the foundation and hold the timbers together. Above B, two timbers C, 14 in. by 18 in. in size, are placed, and between them a series of seven spring timbers D, Figs. 85 and 86. These timbers are full-sized in the middle for one-quarter of their length, and from this point are cut down so that at their ends they are only half this thickness. The shape is shown in the dotted lines of Fig. 87. The ends rest on the iron sills B. The middle supports the bed-plate of the mortar G. By the arrangement of the spring timbers which support the bed-plate of

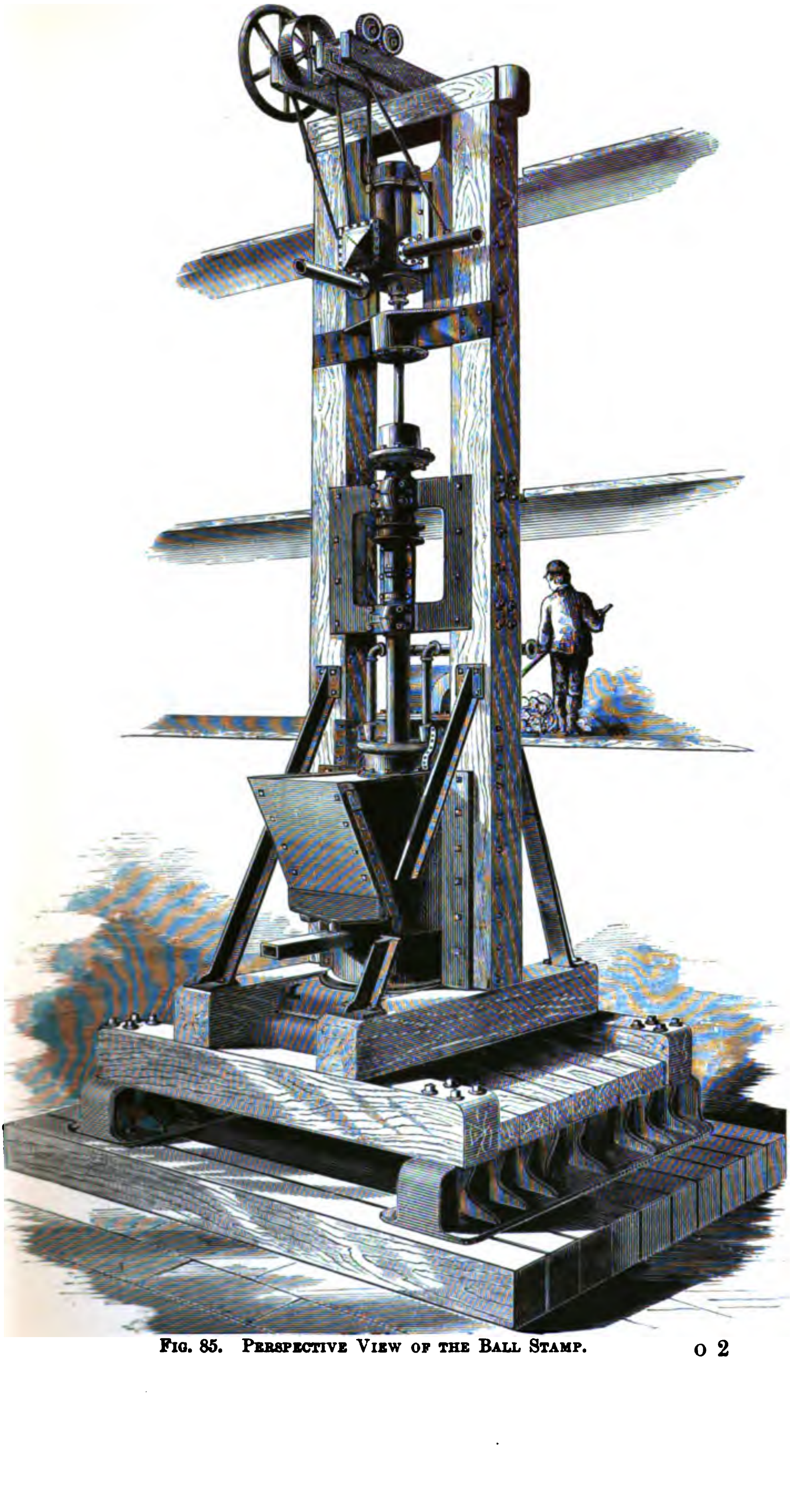


Fig. 85. Perspective View of the Ball Stamp.

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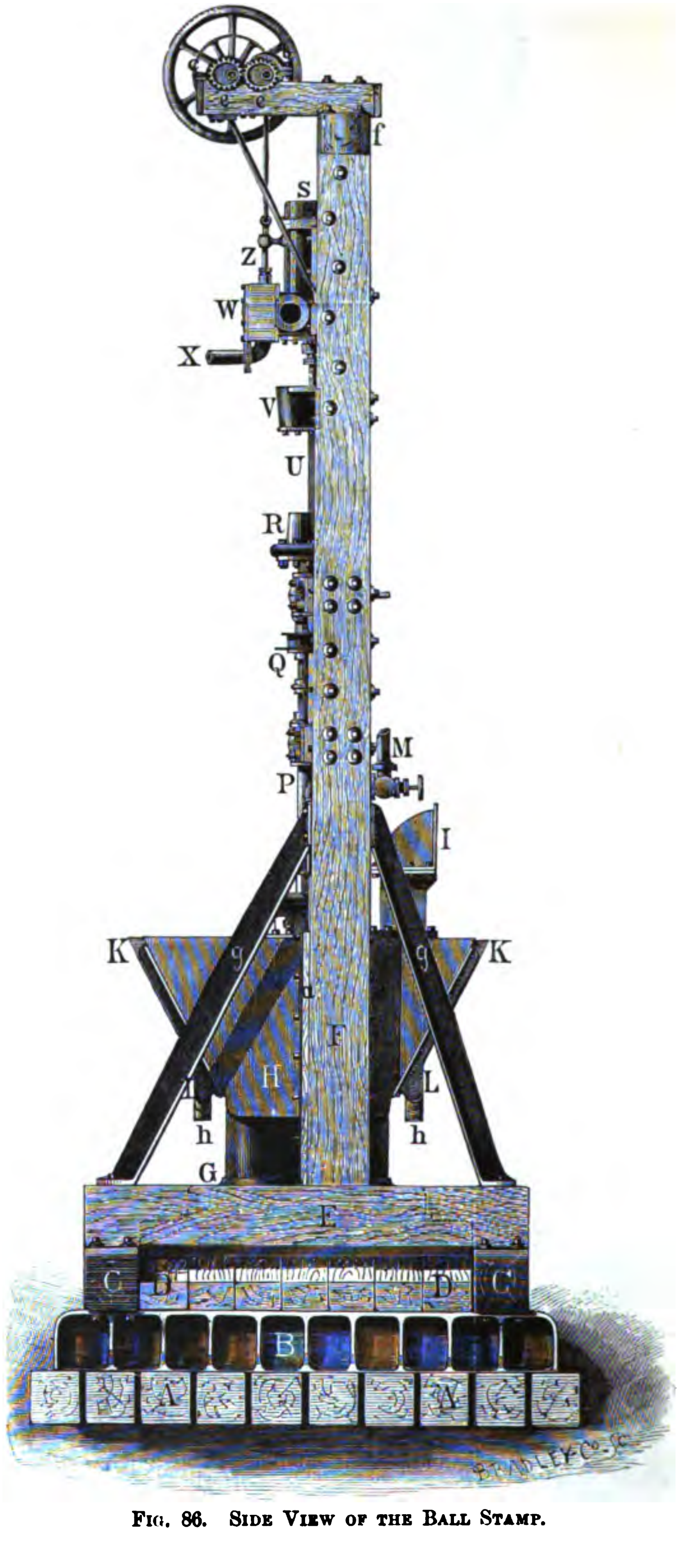


FIG. 86. SIDE VIEW OF THE BALL STAMP.

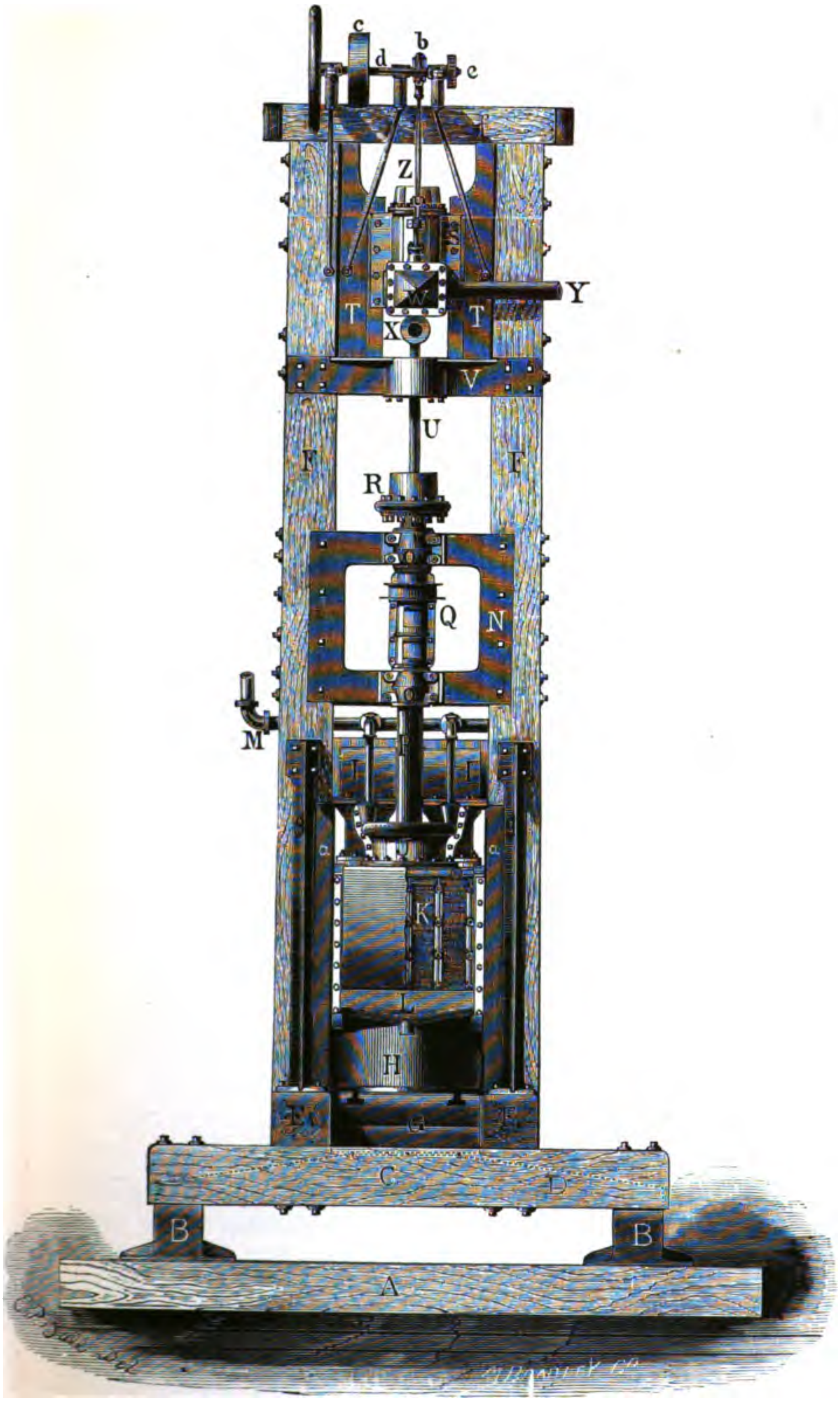


Fig. 87. Front View of the Ball Stamp.

the mortar, the machine and its foundations are relieved from the jar which they would otherwise suffer. These timbers cost $5 each, or $35 a set, and last about one year, when they must be renewed. Oak is considered the best wood for them, and is always used when it can be had; when it cannot be obtained birch is used. On the outside of the spring timbers, and independent of them, is another set of wooden sills E, resting upon and across the sills C, which support the battery posts. These battery posts are held together by a wooden cap *f*, at the top, and by the iron frame N, the bunter beam V and the cheek pieces T T. They are braced by the four iron T beams *g g*.

FIG. 88. SECTION THROUGH THE MORTAR.

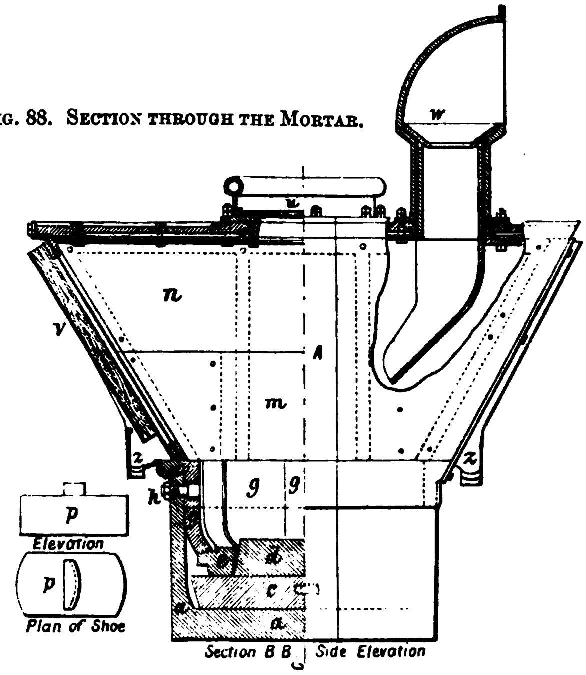
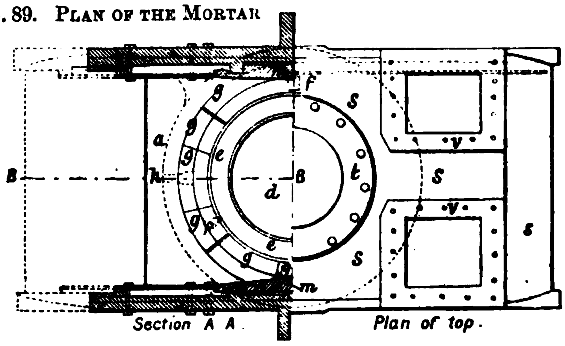


FIG. 89. PLAN OF THE MORTAR



The bed-plate G, which weighs about 3000 lb., is set on the spring timbers between the beams E and secured to them by bolts. On this the lower part of the mortar proper, *a*, Fig. 88, is bolted. It is cast in one piece and weighs 1500 lb. The upper part is generally made with parallel sides, as shown in Figs. 85 to 89, but those used at the Allouez Mill incline away from the mortar, as shown in the plan, Fig. 91. On the mortar bottom the bed-plate *c*, Fig. 88, is placed. It weighs about 2000 lb. On this the die proper, *d*, is placed. This is a circular piece fitted to the bed-plate with a slot. It is from 22 in. to 24 in. in diameter on the top and 24 in. to 26 in. on the bottom, is 7 in. thick, and weighs from 600 lb. to 700 lb. This is surrounded with an iron ring *e*, bevelled at the top, and having three slots, 3 in. by 1½ in., which are circular at their ends and nearly cut out at the bottom for the purpose of fastening the ring to the bed-plate *c* by the bolt *f*. It weighs from 700 lb. to 800 lb. On the edge of the ring the stave linings *g* are placed. These are made of hard chilled cast iron, in sets of nine or more, which are not always of equal size. They all flare on the sides so as to hold each other in place. The last one has a hole through which a steel bolt, *h*, passes, which keeps them all in place; they weigh together about 1500 lb. Still above these are two side linings, *m* and *n*. The stave linings are round. The upper linings have the section of the mortar. The lower side lining, *m*, is generally made of chilled cast iron, and the upper ones of wrought iron. The lower ones are 3 in. thick and 20 in. high, and weigh about 1000 lb. The upper ones are made of wrought iron, and are about half an inch thick; they last about a year on amygdaloid and six to eight months on conglomerate rock. The lower ones last about seven months on soft rock and from four to five on hard. The stave linings last from twelve to eighteen months. They are all changed at one time. The dies last about two years; the length of time depends on the casting. With average castings they last about eighteen months. Usually the whole mortar lining must be changed about once a year, but the parts where the greatest wear comes are always made thick to make them last longer. The parts are always kept in duplicate, so that they can be put in without delay. It takes about four hours to change

the mortar linings. The wrought-iron linings are patched as long as they can be, in order to continue their time of service.

The shoes *p* are made circular, with the sides cut off; on the upper side they have a projecting piece, one side of which is dovetailed to fit into the slot of the stem. The shoes weigh from 450 lb. to 650 lb., and are of hard cast iron made of a mixture of Franklin and German spiegel. It was found that the chilled iron broke too easily. If the chilled iron does not break it wears longer, but it wears rounded, and consequently cannot be used down so close, while the mixture wears much more evenly. It takes from thirty-five minutes to an hour to replace a shoe if the boss, *q*, of the stem is not badly jammed or broken or the shoes broken off in the slot, and sometimes half a day if this is the case. The time depends on how much it has to be chipped to fit it. The shoes wear from five to six days on conglomerate and from two to three weeks on soft rock. With careful work they are rarely broken. In some of the mills they do not break once a year; in others they break once a month. This happens by allowing them to wear too thin, when they break easily. This is a serious accident, for if it should not be noticed at once, and the stem should pound on the die even for a few strokes, it would become so battered that it would have to be taken out, which would involve a serious loss of time. At the Atlantic Mill, in 1875, forty-two shoes were worn out in crushing 8000 tons of rock.

The stem *q*, Fig. 90, will work one or two years without repair, when it must be reforged on the bottom and the shoe slot dressed. It will last ten years before it has to be replaced. Accidents happen to it through the breaking of a shoe, when the stamp gives a peculiar sound, and must then be stopped at once. Generally the shoe slot is not touched. If the shoe does not fit it is dressed.

On the back or feed side of the mortar there is but one screen frame. When the front of the mortar flares there are two screens, each nearly the same size as the back screen, the shape of which is shown in Figs. 91 to 93. The capacity of the screens in the straight mortar was tested at the Sheldon and Columbia Mill by charging the mortar with the stamp sands. It was found that with two men charging, the screen surface was sufficient to

discharge the mortar. The extra screen surface is, therefore, only added when ordered.

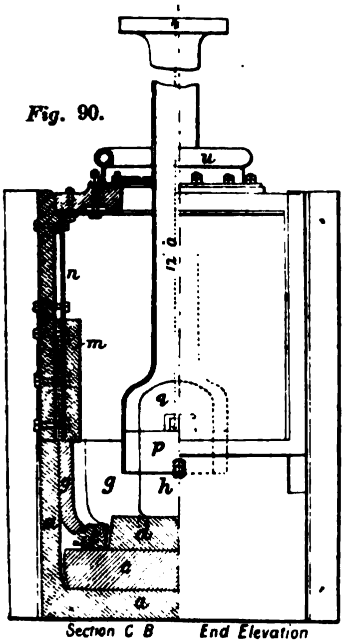


Fig. 90.

The screens are of No. 12 steel plates, $\frac{3}{32}$ in. thick, with holes $\frac{3}{16}$ in. in diameter at the Atlantic and the Calumet and Hecla Mills, and $\frac{3}{32}$ in. at the Allouez, there being about 22,000 holes to a screen. The experiment of using wire screens was made, but they were easily clogged, and wore out rapidly, so that they were abandoned. The holes are made round or oval and are punched by machinery, each mill doing its own work. The oval holes are used because some of the copper has an oblong shape and will not pass a round hole, so that it is in danger of being stamped too fine. Each front

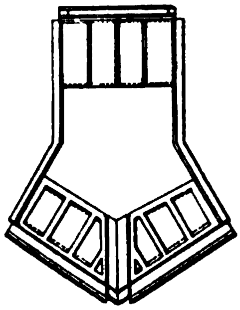


Fig. 91.

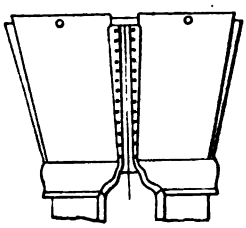


Fig. 92.

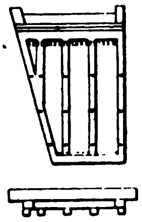


Fig. 93.

sieve frame has four separate steel screens, Fig. 93. The round holes are $\frac{3}{16}$ in. in diameter, and the oblong ones $\frac{1}{8}$ in. by $\frac{1}{2}$ in., or $\frac{3}{16}$ in. by $\frac{6}{16}$ in., which latter is just equal to two round holes.

The screen is clamped between two frames, shown at x in Fig. 88, and is covered on the outside by a sheet-iron plate v, which turns the ore into the discharge pipe z. The inner screen frame lasts two months and the outside one two years. The inside

frame is constantly subjected to the wear and tear of the fine material in the mortar, and consequently wears out quicker. The middle screens last a long time, while the outside ones wear out very rapidly, owing to the constant splash of the stamps toward the outside rather than toward the middle. When the oval holes are used in the screens they are always placed in the outside frames, the object being to let out the flat copper which is too large to pass the round holes. The front and back screens wear out about alike. The mortar lining is changed throughout about once a year, but the parts where the greatest wear comes are made double.

When a screen is cracked and it is not desirable to stop the stamp long enough to repair it, the crack is filled up with a wedge of wood. When the front screen is to be removed it is lifted off with a block and tackle, a hole being made in the top of the screen frame for this purpose. It is slid into its slot again over a flat bar, and is bolted overhead with two bolts. The back screen holders are the same as the front ones, except that they are T-shaped, and have on each bar four 1½-in. rounds commencing 4 in. from the bottom and top and 1 ft. apart. The back screen holders are held in two frames. They are 38 in. wide and 47 in. long. The top is 3 in. wide, the separation 2½ in.; the screens are of equal size. It is very easy to remove the front screens, but to remove the back ones the floor where the feeder stands has to be taken up, and they are more difficult to move, as the space is much more contracted.

At the Atlantic Mill the screens of the stamps originally had ¼-in. holes. They then used $\frac{3}{16}$-in. holes for three months, and then ⅛-in. With these last holes there was a great loss, as the copper was pounded too fine. With ¼-in. holes there was a loss from the large size of the rock. When they went back to $\frac{3}{16}$-in. holes the product of the mill came up at once. The front screens last three weeks. The back screens are taken out when they are worn. The breaks are generally in the bottom. The screens are turned upside down and put in again on the back side, where there is less wear than on the front. When they can no longer be used as screens they are used in some mills to line the bottom of the launders which bring the water to and carry off the tails from

the mill, as at the Calumet and Hecla; in other mills they are thrown away.

The screens discharge into hoppers L, Figs. 86 and 87, and *z*, Fig. 88, attached to the inclined sides of the mortar, and deliver the material to launders which carry the crushed ore to the washers. The top of the mortar is closed; on it is the feed hopper I, Figs. 86 and 87, and W, Fig. 88, which is bolted to the planed surface *v*, Fig. 89, on the top of the mortar, and must be changed about once in three months. The stem of the mortar passes through an urn-shaped appendage, J, Fig. 87, and *u*, Fig. 88, with a cone in the centre, through which water is discharged into the mortar by the pipe M, Figs. 86 and 87. Just above the mortar an iron frame N, Fig. 87, is bolted to the battery posts F, which has two boxes O O, through which the stem of the stamp P is guided. Between these two boxes a pulley with feathers Q is clamped, which works in splines in the stamp stem; a belt running over this pulley gives a rotary motion to the stamp. The stamp stem P, is round, made of wrought iron, Figs. 87 and 90, and is finished throughout. Its lower end has a dovetail to receive the shoe, which is fitted and keyed into it. The upper end has a circular flange to which the bonnet R is bolted. At the top of the mortar frame the steam cylinder S is bolted to a cast-iron frame T, which is fitted to and bolted to the stamp frame. The piston passes down through the bunter beam V, which is bolted to the battery timbers, into the bonnet R, into which it is screwed, as shown in Fig. 87. The bunter beam frame contains a cushion against which the top of the stamp shaft bonnet R would strike if it were raised too high, and this prevents its upward motion, and also prevents the piston from knocking the cylinder head out. It never touches except when too much steam is let into the steam cylinder, or when the pressure of the boiler is unnecessarily increased. The steam chest W incloses the slide valve; the steam enters the steam chest W by a pipe Y, and is discharged by the exhaust pipe. The slide valve works entirely independent of the stamp stem, running a fixed number of strokes per minute, and is driven by the eccentric *b* and valve rod Z. The eccentric has its motion from a belt on the pulley *c*, which is driven by a separate engine. It runs upon the shaft *d*, upon

which are two eccentric elliptical cog-wheels *e*. The regular motion of the shaft *d* is in this way changed into an irregular motion, and gives the eccentric and steam valve a slow upward and quick downward movement. By the arrangement of cushions in the stamp shaft bonnet R, the piston-rod and head are relieved from the very severe strain which would otherwise come on them. The mortar of this stamp is so large that the shoe and die must never come together, as the power of the blow is very great. A lever is attached to the side of the stamp which the bonnet R strikes when the rock gets too low, and warns the workmen before the shoes and dies come together.

To work the stamp the motion is communicated to the valve by the pulley *c*, the ore having first been thrown into the feed hoppers. It falls down into the mortar proper through the feed spout I, Fig. 86, and W, Fig. 88. The water is turned on to the pipe M, Figs. 86 and 87, and at each blow of the stamp the water and crushed ore are thrown toward the top of the screens. The pulp is forced through the screens and falls through the spout *z*, Fig. 88, into the launders leading to the washers. The stamps work generally twenty-six days in a month. When a stamp needs repair there should be a spare stamp to use in its place, and it should be immediately repaired. At the Atlantic Mill this system is adopted for the boilers also.

The steam cylinder of number one stamp is 12 in. in diameter and has a 20-in. stroke. The two medium-sized stamps at the Allouez Mill make from 95 to 100 strokes per minute; the latter number of strokes will break the most rock. It stops only half an hour a day. In the month of August, 1876, they crushed at the Allouez Mill 3320 tons in twenty days, or 83 tons a day for each stamp. At the Calumet and Hecla, with seven stamps, they crush 770 tons in twenty-four hours. At the Atlantic Mill the large-sized stamp makes 85 strokes a minute, and each head crushes 110 tons, or 330 in the mill in twenty-four hours. Since 1876 the capacity of the stamp has been increased to a maximum of 165 tons of rock crushed in twenty-four hours.

The steam cylinder of the stamp must be occasionally rebored. If the casting is soft and the stamp constantly watched, it may have to be rebored once in six months. If hard it will last

according to the care taken of it, from one to two years. At the Atlantic and Allouez Mills they rebore once a year; at the Calumet and Hecla once in two years. They cannot generally be rebored oftener than three times; after this they must be replaced. The piston-rod lasts from twelve to eighteen months, when it must be renewed. The piston-rods, valve stems, &c., are all made to a size.

At the Atlantic Mill the steam goes to the stamps at a pressure of 95 lb. The exhaust steam from here is run into a receiver and heater 50 ft. long and 2 ft. in diameter, the steam pipe from which goes to the main mill engine. The mill engine is thus run entirely by the waste steam, and the water for the boilers is heated. They burn 14½ tons of coal per twenty-four hours and stamp 330 tons of rock every day. The coal costs about $4.50 delivered at the mill. The water is brought in launders 20 in. by 20 in. and 2½ miles long. During the winter they are covered tight. The quantity of water used per minute is 220 gallons. This is about 25 tons of water for one ton of rock. At this mill, on account of the faulty construction of the pockets, two men are required to feed the stamps. At the Allouez Mill there is but one feeder, except when the rock is very low, and a man has to go into the bin to throw a supply to the mouth. At the Calumet and Hecla there are three chargers per stamp. This is necessary on account of the faulty construction of the bins, which were made to hold three days' supply of rock, which is necessary to guard against any accident to the railroad. When the supply of ore in the bins gets low on account of stoppage of the ore trains, two men have to go inside and shovel it out.

There is one man at the valve for two stamps, and one man changing screens and fitting the shoes. The shoes require a great deal of work, as they have to be shifted and fitted. They come to the mill exactly as they came from the foundry. The man who looks after the screens attends to these also. There are eight firemen for the boilers of the stamps; there are two blacksmiths. In their machine shop there are ten men; these do all the repairing and new work as well. All the repairs are done on Sunday.

At the Allouez there are three men for twenty-four hours to a

stamp, who work eight hours each, and two engineers in twenty-four hours. There are two men who regulate the steam for twenty-four hours. At the boilers are six firemen. There are in the machine shop one blacksmith and two machinists. Wood in August, 1876, cost 33 cents per ton of rock stamped. The two stamps used 250 cords at $2.50 per cord. This wood is all cut on the company's lands. The boilers were not in good condition and used too much wood. The pressure of steam is 100 lb. There are three 2-in. water pipes to each head. The quantity of water used per day for all purposes is 1556 gallons. It is brought in launders 24 in. by 16 in., for both winter and summer supply.

At the Atlantic Mill the water is supplied by two 2-in. gas-pipes, the supply for the mill being 30 to 32 tons of water for each ton of rock dressed.

The patterns for all parts of the stamp liable to get out of order are kept in the shop, so as to allow of immediate repair when necessary. When any part wears out it goes to the old iron. There are two men constantly engaged in repairs to the mill. Only the best mechanics should be employed at repairs. At the Atlantic Mill they are paid $74 per month; at this mill there is one blacksmith.

The following Table, prepared for me by Mr. E. J. Murphy, of the Hartford Foundry, gives the weights and cost of the principal parts of the Ball stamp:

Cast Iron.	Weight—Lb.		
2 sills or girders	... 11,730		
1 bed-plate (in two pieces) ...	... 21,895		
1 mortar	... 15,300		
1 stamp shaft guide cap and box	... 2,634		
1 bunter beam and ring ...	... 1,206		
2 feed hoppers and attachments	... 1,473		
2 discharge hoppers	... 678		
2 screw frames	... 361		
1 urn	... 480		
4 braces for stamp frame ...	... 2,500		
2 cheek pieces	... 876		
1 set of linings for mortar ...	... 1,622		
1 bottom piece for mortar ...	... 695		
1 stamp head die...	... 550		
1 ring	... 800		
1 steam cylinder and attachments	... 2,722		
		65,522 at 6 c.	... $3,931.32
Brass.			
Composition valves and bushings	...	64 at 50 c.	32.00

	Carried forward ...	$3,963.32
Sundries.		
Boiler plate	1,692	109.48
Wrought-iron bolts, nuts, keys, &c. ...	2,228 at 8 c.	178.24
Steel for keys	81 at 25 c.	20.25
Eight steel screen plates	100	18.00
Steel piston rod	130	20.75
Steam piston head and follower ...	173	37.11
Rubber springs	... ...	125.40
Two flanged cocks	... ...	28.00
Timber.		
2 sticks, 27½ feet by 18¼ inches by 18¼ inches; 2 ,, 11½ ,, 18¼ ,, 18¼ ,,; 1 ,, 9 ,, 12¼ ,, 16¼ ,,; 3 ,, 5⅓ ,, 4¼ ,, 10¼ ,,	2,377 feet at $70.	185.50
Paint, nails, lumber and boxing... ...	... ...	32.72
Boxing and patterns	... ...	50.00
Carpenters making frame	... 34 days at $5.00.	170.05
Labour by machinists	...5,488 hours at .50.	2,744.00
,, blacksmith	... 221 ,, 1.25.	275.70
Cartage	... ...	41.48
Stamp Shaft.		
1 stamp shaft	... 3,365 lb.	412.33
Labour by machinist	... 229 hours... ...	114.50
Lathe work	... 171 ,, ...	256.00
Total	... ...	$8,782.83

The following Table shows the operations of the Pewabic Mill:

PEWABIC MILL, 3 Heads Ball's Stamps, January 1 to December 31, 1875.

OPERATION OF MILL.

——	No. of Cords of Wood used.	No. of Tns. of Coal used	Cost of Wood.	Cost of Coal.	Cost of Supplies, Oil, Packing, Belting, &c.	Cost of Foundry Bills.	Cost in Wages.	Total Running Expense.
			$	$	$	$	$	$
January .	484	...	1,815.00	...	361.85	204.81	1,862.86	4,244.52
February .	477	...	1,788.75	...	278.00	141.73	1,445.13	3,653.61
March .	510	...	1,912.50	...	306.31	181.57	1,716.38	4,116.76
April .	380	70	1,425.00	420	282.01	173.06	1,606.88	3,906.95
May . .	425	...	1,593.75	...	275.99	194.30	1,772.56	3,836.60
June .	452	...	1,695.00	...	199.24	209.17	1,643.40	3,746.81
July . .	455	...	1,706.25	...	212.80	60.88	1,754.49	3,734.42
August .	485¼	...	1,819.69	...	175.12	138.36	1,695.99	3,829.16
September	428¼	...	1,605.93	...	263.55	144.65	1,596.03	3,610.16
October .	404	...	1,313.00	...	514.12	148.58	1,632.66	3,608.36
November	429	...	1,737.45	...	453.79	251.00	1,617.60	4,059.84
December.	403	...	1,632.15	...	449.30	173.57	1,606.16	3,861.18
Total .	5332½	70	20,044.47	420	3772.08	2021.68	19,950.14	46,208.37

RESULTS.

——	Number of Days Running.	Tons of Rock Stamped.	Tons Stamped per Cord of Wood.	Cost of Stamping and Washing One Ton of Rock.
January . .	24	4,762	9.83	$0.89
February . .	21	4,355	9	.84
March . .	22	4,638	9	.88
April . . .	24	4,729	9.75	.82
May . . .	24	5,028	11.80	.76
June . . .	24½	5,033	11.13	.74
July . . .	25	5,261	11.56	.71
August . .	24½	4,952	10.21	.77
September . .	24½	5,260	12.29	.68
October . .	24¼	5,062	12.53	.71
November . .	23	4,819	11.23	.84
December . .	21	5,043	12.51	.77
Total . .	281¾	58,942	Average 10.84	Average $0.79

The following Table shows the operations of the Allouez Mill:

ALLOUEZ MILL, 2 Heads Ball's Stamps, in 1875-76.

——	No. of Cords of Wood used.	Cost of Wood.	Cost of Supplies.	Cost of Foundry Bills.	Cost in Wages.	No. of Men employed.	Total Running Expense.	Tons of Rock sent to Mill.
1875.		$	$	$	$		$	
July . .	500	1,250.00	289.46	632.80	1,672.12	40	3,844.38	*2,205
August .	855	2,137.50	826.60	213.60	1,898.32	44	5,076.02	4,096
September	784	1,960.00	344.11	570.69	2,021.83	49	4,897.63	4,656
October .	790	1,975.00	301.91	614.60	2,361.33	48	5,252.84	4,450
November	625	1,562.50	311.10	580.92	1,741.67	38	4,196.19	3,600
December.	730	1,810.00	474.90	423.86	2,255.63	45	4,964.39	4,976
1876.								
January .	826	2,065.00	362.33	384.45	2,190.39	48	5,002.17	4,200
February .	620	1,550.00	238.63	662.32	2,314.43	47	4,765.38	†2,500
March .	830	2,075.00	136.31	318.00	2,084.62	46	4,613.93	4,812
April .	753	1,882.50	155.15	285.82	1,673.94	40	3,997.41	5,295
May . .	830	2,075.00	251.45	376.25	1,774.93	44	4,477.63	5,015
June. .	760	1,900.00	282.07	435.90	1,740.30	43	4,358.27	5,330
Total .	8903	22,242.50	3974.02	5499.21	23,729.51	532	55,446.24	51,135

* Short of water from drought.

† Short of water from the launders freezing solid.

RESULTS.

—	Tons Stamped per Cord of Wood.	Pounds of Copper produced.	Cost of Stamping One Ton of Rock.
1875.			$
July . . .	4.40	80,430	*1.74
August . .	4.79	170,970	1.24
September . .	5.90	180,050	1.05
October . .	5.63	170,055	1.18
November . .	5.76	129,680	1.16
December . .	6.91	200,350	.99
1876.			
January . .	5.08	160,054	1.34
February . .	4.03	90,440	†1.90
March . .	5.80	160,460	.96
April . . .	7.00	172,323	.76
May . . .	6.00	180,130	.89
June . . .	7.00	180,455	.82
Total . .	68.30	1,875,367 or 937.698 tons.	$14.03

Expenses of saw mill and repairs to railroad included.

The following Table gives the sizes and dimensions of the Ball stamps:

List of Regular Sizes.

No. of Stamp.	Diameter of Shaft or Stem, in Inches.	Weight of Stamp Shaft and Shoe, in Pounds.	Extreme Stroke or Lift, in Inches.	Diameter of Steam Cylinder, in Inches.	Number of Blows per Minute.	Horse Power required for One Head.	Actual Amount of Rock Crushed per Day of 24 Hours.
1	8	4500	28	12 to 15	90	60	120
2	7	3500	28	11	90	52	96
3	6	2500	26	9	95	35	65
4	5	1500	24	8	110	16	35
5	4	650	18	6	120	8	15 to 20

Mr. E. D. Leavitt‡ has made an improvement on these stamps by which the energy of each blow has been increased to 40 foot-tons. The stamps are capable of crushing 225 tons each of conglomerate, this being the average duty of a two weeks' run. They consume, per ton of rock crushed, 40 per cent. less fuel than was formerly used.

* During this month the men were on hand, but the mill was short of water.

† Short of water from the launders freezing solid.

‡ Trans. Am. Soc. Mech. Eng., vol. vi., p. 380.

ROLLS.

Up to the year 1882 all the efforts made to crush ores fine were directed toward the improvement of the California stamp. When it reached its maximum efficiency, however, it was still found that something was lacking, not only because of the limited production per stamp head, but also because of the cost and the wear and tear of the stamps themselves. Until about that time it had been supposed that the California stamp was the perfection of machinery for crushing, and that nothing whatever could take its place. It was admitted that rolls were suitable for crushing ore to such a granular state as is most suitable for concentration, but even in concentration works, the stamp was largely used. The prejudice against rolls arose in this country mostly from very ill-advised experiments, undertaken in the early days of Lake Superior, when the effort was made to reduce hard rock, containing large nodules of native copper, to a condition fit for concentration, with rolls made of cast iron. Even these might have been successful but for the ill-judged economy of making the machinery so light that it was hardly strong enough to crush the rock, and entirely incapable of rolling out native copper, so that the copper lumps could not pass the rolls, which were consequently often jammed by them, so that they had to be taken apart, if the force was not sufficient to roll them out, or were broken if it was.

In 1883, the Bertrand Mining Co., of Nevada, influenced by the success of Krom's geared rolls in the leaching works in Galena, Nevada, introduced his improved steel rolls for crushing. It soon became evident that the dictum which had been received everywhere that rolls could not be used for crushing in the metallurgy of gold and silver was a mistaken one. It was soon seen not only that the cost of the plant was much cheaper, but also that the

efficiency, so far at least as leaching work was concerned, was far greater, for the pulp which came from the rolls was found to contain much less fine dust, and the particles to be much more uniform in size than those which came from the stamps. It had before been supposed that an impalpable powder was necessary in order to have all the precious metals in the ore transformed into chloride, where the ores had to be roasted, or so that the mercury could reach the gold and silver in the ore when it was free milling. Recent practice, however, has shown that for chlorurising roasting, excessive fineness of ore is not necessary, and that it may be even injurious in lixiviation, by interfering with the rapidity and efficiency of the filtration.

The success of the rolls in the Bertrand Mill induced others to adopt them. The results have been so far favourable, but it is impossible, on account of the short time in which they have been in use, to make an exact comparison between their efficiency and that of stamps. Enough, however, is already known to be certain that in every respect the comparison is in favour of rolls, though to what extent cannot at present be accurately determined. It is known now that two sets of rolls, from 14 in. to 16 in. long and from 26 in. to 30 in. in diameter, run at a velocity of 100 turns a minute, are equal to a 50-stamp mill, where the stamps weigh 850 lb., and that the amount of fine dust produced by rolls is not as great as that produced by stamps. When it is taken into consideration that a 50-stamp mill involves a very large and expensive house and foundations, while the rolls are self-contained and require but a small space, the very great advantage of the rolls is at once apparent, independent of the original cost of the machinery, which is much less. There seems to be no more difficulty to crush fine with properly-constructed rolls than with stamps. The capacity of rolls, like stamps, is limited by the screening capacity and by the fineness of pulverization. At the Austin Mills, in Nevada, it was found that crushing through a No. 16 screen, there were produced 16 per cent. of fine material which passed a No. 100 screen, but only half of this, or 8 per cent., went to the dust chamber.

The growing poverty of the old mines, the uncertainty of the discovery of new ones, the invention of new processes, by which

poorer ores can be treated, and the desirability of reducing expenses in all the various stages of producing the metal, has turned attention to rolls. The old Cornish rolls are not adapted to a large output, and are more expensive in labour than the ores of the West justify. Many attempts have been made to replace them by crushers, disintegrators, and pulverizers, many of which are ingenious, though some of them are quite complicated, but none of them solve the question of a maximum of output produced by the use of a minimum of power, repairs, and labour.

Of the various kinds of improvements in rolls which have been made, the most ingenious, and at the same time the most simple in construction, involving the least labour and the least cost in repairs, are the rolls made by Mr. S. R. Krom, of New York, which it is our purpose to describe, and to compare with stamps both for economy of construction and repairs, and increase in efficiency and output. These rolls are composed of two sets of cast-iron cores B on steel axes C C^1, Figs. 94 and 95, carrying steel tyres A. The rolls vary from 26 in. to 30 in. in diameter, including the steel tyres. These tyres are made of the best open-hearth steel and are 2½ in. thick on the 26-in. rolls, and 2¾ in. thick on the 30-in., and can be worn down to ½ in. They have been worn as low as ¼ in. The tyres are used until they are so thin that they become loose from expansion. The pillow-blocks L of one of these rolls is firmly bolted to the bed-plate F by the nut N. The second one is set in a swinging pillow-block fixed in two strong cranks D D^1, which rotate in a journal E set in the cast-iron frame F. The shaft M, which connects the two swinging pillow-blocks, is 11 in. in diameter, so that this roll and shaft are always held in line with the other roll. When rolls are constructed with pillow-blocks which slide on the bed-plate independent of each other, it is difficult to keep the two shafts parallel; besides there was always a liability of the movable pillow-blocks getting loose on the bed, which is not only objectionable, but causes damage to the bottom face of the pillow-block, and the face of the bed-plate. But with this construction all the pillow-blocks are securely fixed. One pair is firmly bolted to the bed, while the movable pillow-blocks swing on the strong pivot E. The distance between the two rolls is

Fig. 94.

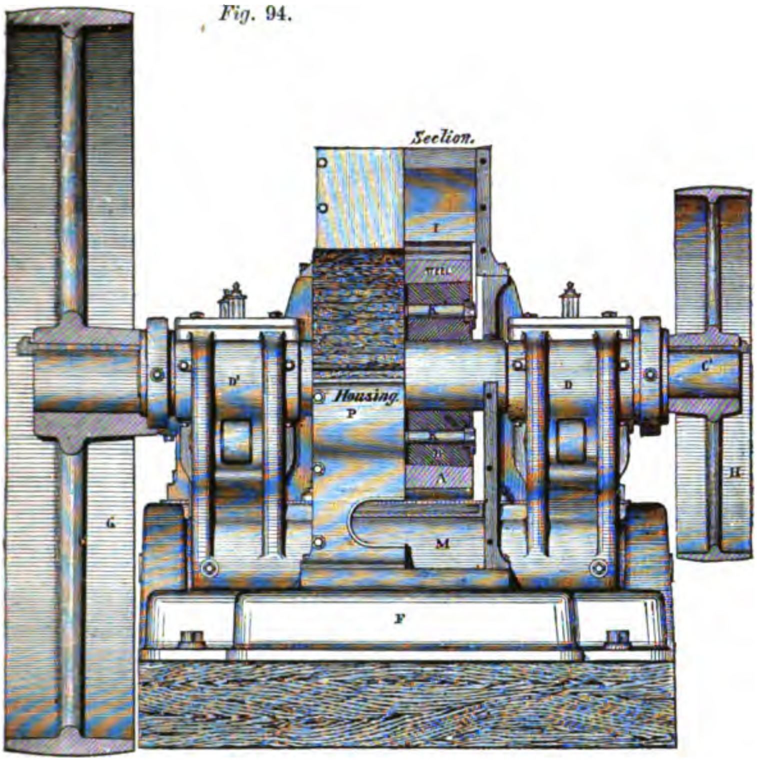
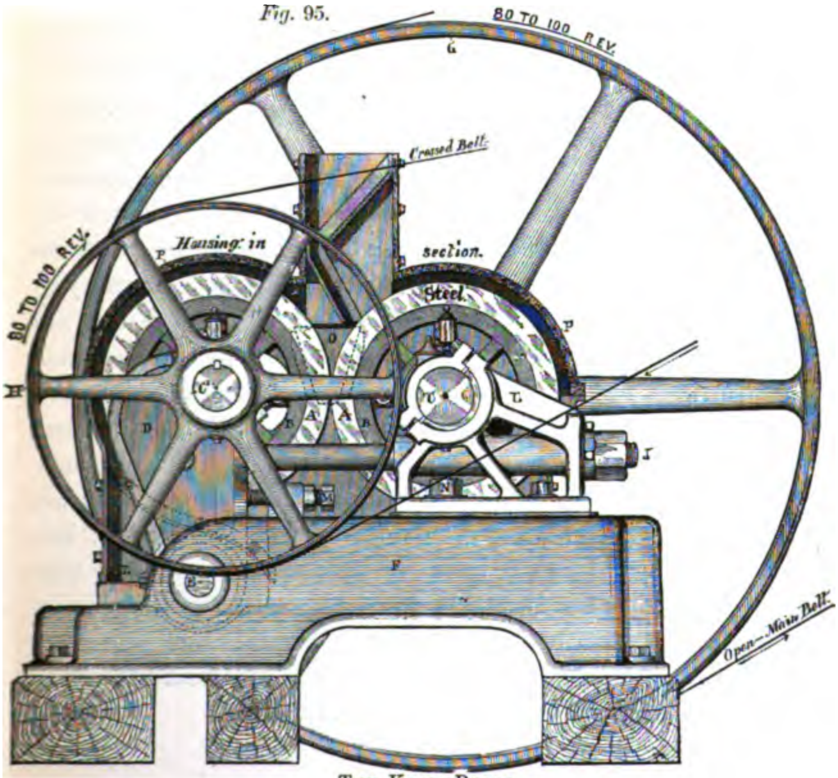


Fig. 95.



THE KROM ROLLS.

regulated by the screws M, one on each side, with their jam nuts to prevent any motion after they are once set. The rolls are held by two heavy bolts J, so that the position once fixed, the distance of the surfaces cannot be changed by any action of the machine, but only by the wear of the rolls. To the fixed rolls a large driving wheel G, 7 ft. in diameter and 15 in. wide, is keyed on its axis C, and to the movable one a smaller wheel H, 42 in. in diameter and 8 in. wide, is keyed on its axis C'. The rolls are covered with a housing P, to which an exhaust fan is attached, so that no dust escapes into the air, the whole of it being carried to the dust chamber. To this housing a feed box I is attached, which is provided with a series of inclines, so as to spread the ore in a continuous and even sheet between the surface of the rolls. Magnets are attached to the feed chute to catch any pieces of hard steel which may have fallen into the ore from broken tools, which might dent the surface of the rolls; iron will do no injury to the faces. The whole machine is self-contained and very compact, a pair of 26-in. rolls occupying a ground space of 7 ft. by 7½ ft. The tyres, which for the 26-in. roll weigh 816 lb. each, are held in place by two cast-iron heads B, which are slightly conical in shape. One of these is shrunk on the shaft, the other is slit on one side and slips on to it. Both of the heads B are so placed on the shaft that the smaller diameter will be toward the centre. The steel tyre is turned out on the inside to correspond to this, so that it can be easily slipped over the permanent head and the loose one brought up to it. The two are securely fastened together by bolts K, so that when the movable head is drawn up to the permanent one, the slit in it closes, and makes it perfectly tight on the axle. It was thought at first that the steel tyres would wear unevenly, but they do not. When the tyres are worn thin, they become loose; they can then be very easily removed and others substituted. To prevent loss of time it is a good plan to have duplicate rolls, so that there may be no delay when the tyres are worn out, as the putting on of a new set requires but a short time. Movable cheek pieces are placed at each end of the rolls to keep the ore from spreading sideways. These cheek pieces are also adjustable.

It has been said that any accident to the rolls would involve the stoppage of the whole mill, while in the case of stamps, only one battery of five stamps would be hung up. This objection, however, falls to the ground, for it has been found that accidents to the rolls are extremely rare, and that repairs can be very quickly made, while the power is much more advantageously used than in stamps, besides this the greater capacity of rolls has led to the use of storage bins in which to accumulate pulverized ore, so that the rolls are run but twelve hours per day.

The rolls are usually arranged in sets of two. One of these receives the coarse ore from the crusher, which, after passing them, falls on a screen of determined size. What does not pass goes to a second pair. What passes the screen goes to treatment. When very fine crushing is required three pairs of rolls can be used, but generally two sets are sufficient. The use of band wheels to drive the rolls in the place of gearing, as formerly used, not only simplifies the machine, but allows the rolls to be run at a higher speed, and they are therefore capable of doing more work. The bolts which take the strain and the swinging pillow-blocks which keep the rolls parallel secure a uniform product. The arrangement of the hopper for spreading the ore evenly across the faces of the rolls, and the ease with which the tyres can be renewed, secures the least wear for the greatest output. The method of drawing off the dust by a fan protects both men and machinery from injury. The space occupied by the rolls is exceedingly small for the work done. Below is given the weights of the different parts of the 26-in. and the 30-in. rolls.

	lb. 26 in.	lb. 30 in.
1 stationary pillow-block	400	500
1 ,, ,, ,, ...	500	600
1 swinging ,, ,, ...	1700	2090
2 rolls with their shafts	4400	6400
Housing with hopper	700	800
7-ft. driving bandwheel	1700	—
8-ft. ,, ,, ...	—	2000
42-in. pulley	333	—
48-in. ,, ...	—	370
Bedplate	3400	4730
Forgings, bolts, &c.	1086	1120
	14,219	18,610

The rolls are made of three sizes, with varying lengths and diameters. These are 22 in., 26 in., and 30 in. in diameter, and 14 in., 15 in. and 16 in. long. It has been suggested that they should be made longer, but there is no practical advantage in making them so. In considering their efficiency and capacity, we must take into consideration that the whole of the surface of the faces of the rolls is fitted to act on the ore, since they are even and parallel and wear so. The ore escapes from the rolls by gravity just as soon as it is crushed fine enough to fall through the space between them. If the feeding is automatic, or even when it is not, there can be no clogging of them possible. In order to compare them with stamps, the surfaces actually in contact at any given moment must be compared. A 30-stamp mill, with stamps weighing 850 lb. dropping ninety times a minute, with shoes and dies 8 in. in diameter, which have, in round numbers, 50 square inches of surface, will have $50 \times 90 \times 30 = 135{,}000$ square inches of surface acting on the ore every minute. It may be considered that two sets of 22-in. rolls will have an average diameter of 21 in. If these rolls make 80 revolutions per minute, and are 14 in. face, they will have a contact surface of 141,120 square inches per minute, or a little more than a 30-stamp mill. If the number of revolutions is increased to 100, the capacity is brought up to 171,000 square inches of surface, a little less than a 38-stamp mill, whose capacity is 176,400 square inches of surface. If the diameter of the rolls is increased to 26 in., with 15 in. of face, the average diameter may be considered as 24 in. If they make 80 revolutions a minute, the face surface will be 162,800 square inches, equal to the surface of a 36-stamp mill. If the number of revolutions of this same roll is increased to 100, the surface capacity is the same as that of a 48-stamp mill. If the diameter is increased to 30 in., taking the average as 28 in., and the length of face 16 in., and the number of revolutions 80, the surface capacity will be equal to 47 stamps, and at 100 revolutions to 60 stamps. If in each of these cases where two sets of rolls have been considered, a third set is added, the capacity will be increased 33.3 per cent., and if four sets are used, doubled. These calculations are made on the supposition that

the surfaces acting are of equal efficiency in both stamps and rolls, and are for that reason more favourable to the stamps than actual practice shows. With rolls properly constructed the pressure is constant at each instant of time, while with stamps, on account of the varying height of the ore in the mortar, the cushioning of the stamp against the ore, and the fact that the stamp must not only crush, but also force the ore through the screens, it never can be constant. Actual practice in the mills has shown that the output of the rolls is always larger than that given by these calculations. That the work can be done as well with small rolls, which are very compact and take up only a little space, as with stamps which occupy a much larger one, is shown by the experience of the mills where they are used; that they can crush as fine is shown by the fact that rolls are now being used to crush after the stamps.

Formerly the power was applied to the rolls by gearing, which was constantly liable to get out of order, and limited the speed at which it was possible to run. Pulleys are now used, a belt passing over both the large and the small wheel. This allows of attaining a speed of 80 to 100 revolutions a minute, or higher if desirable, a speed entirely impracticable with gear. The capacity of the rolls is thus increased, while the wear is confined almost entirely to the crushing faces. It does away with the noise of the gearing and reduces the risk of breakage almost to nothing. Both rolls, when the machine is in operation, travel with the same speed of surface, but the small one, H, is so speeded that when there is no ore between the rolls it travels one or two revolutions per minute faster than the large one. The reason why this double system was adopted is that one of the rolls being movable and the other stationary, it would not be a good construction to place a large pulley on a movable pillow-block. When ore is between the rolls a single actuated roll will cause the other to revolve, so that if most of the power to do the work is applied to the large wheel only a small part of it will be necessary to insure that the other roll will always bite when fed with ore and be kept in motion if the ore should for an instant fail. It will be seen also that the strain on the rolls is taken upon the bolts J, and that by the use of the two bolts J and M with the nuts N, all of

which are provided with double nuts, no pounding action is possible. The defect of all the other rolls has been that they did not provide against this pounding movement, and that they have been too light in weight to do the duty required of them. Both of these defects have been remedied in this machine. The surprisingly small number of parts, all of which are easily accessible, their simplicity, as well as the small space occupied for such a very large output, and the possibility of adjusting the distance between the rolls at any time in a very few minutes, makes this machine, which is entirely self-contained, not only a most efficient, but a very economical one.

These rolls are now in successful operation at the Bertrand and Mount Cory Mills and Wenban's Mill, Cortez, Nevada, and they are being introduced in other mills. It has been shown that the two sets of 26-in. rolls at the Bertrand Mill can easily crush 150 tons of hard ore in twenty-four hours so as to pass a 16 screen. How much more they can do is not known, as they have never had full screen and elevator capacity. The Mount Cory Mill has crushed 100 tons in the same time through a 30-mesh screen. The capacity of the best stamps on the same kind of ore would be about 2 tons per stamp, which shows the rolls equal to a 50-stamp mill. At the Bertrand Mill 9000 tons of ore were crushed with no outlay for repairs, with the expenditure of less than one-half the power required to do the same work in a stamp mill, while the total expense for wear has been only about one-quarter of that of a stamp mill. At this mill 15,000 tons of ore passed the fine crushing rolls before new tyres were necessary. After crushing 20,000 tons the coarse crushing tyres were still good for two months wear. They consider that each set of rolls with a set of repair linings of composition metal will be capable of crushing 20,000 tons; the only expense for repairs being the tyres and the cheek pieces. The cost for the steel for a set of 26-in. rolls during 1885 was:

Two sets of tyres	$524
Freight on 3204 lb. at $0.03	98
Composition liners for journals and cheek pieces ...	100
	$722
Or per ton of ore treated	$0.0361

To make a comparison between the efficiency of rolls and stamps the possible output and the cost of doing the work must be considered. The quantity of material which can be treated in any given time will depend on the amount of actual crushing surface which is in contact at each moment of time. It is evident that that machine will be the most efficient which presents the greatest amount of such crushing surface. The cost depends on the amount of power required to do the work and the expense of repairing the wear and tear of the machine and keeping it up to its greatest efficiency. The cost will be greatest when any part of the power is badly expended or wasted, and when the number of parts to the machine is so large as to make the construction or repairs expensive, and when for any reason skilled labour is difficult to obtain. For this comparison we will suppose that the rolls have been used and are worn down to a diameter which will represent the average wear of a 26-in. roll, and that the rolls are 15 in. long, running 100 revolutions per minute. The crushing surfaces of two such sets of rolls in actual contact at every instant of time will, as we have seen, be equal to that of a 50-stamp mill with shoes and dies 8 in. in diameter, making ninety drops per minute. The action of the rolls is continuous. They have nothing to do but crush the ore which falls between them. If during any instant of time there is no ore, the power is stored up in the fly-wheels to be used the next moment. The power of the stamp is spent in first crushing the ore and then forcing it out of the screens. As the whole of it cannot be forced out at every blow, a considerable part of it will be acted on several times without passing the screen, and this ore not only consumes the force of the blow of the stamp, but cushions it, and thus still further reduces its effectiveness. The weight of the stamp and the height to which it must be lifted is a constant quantity, hence the power required to move it will be constant, while the effectiveness of the blow depends on the size of the ore from the crusher, the quantity of it in the mortar, and the effectiveness of each blow to force the crushed material out. No careful experiments have as yet been made to find the actual amount of power exerted in a stamp mill to crush the ore only, but the stamp has so much else to do that it is safe to say

that it does not exceed 50 per cent. of the power actually expended. With the rolls held rigidly together the action is constant at any given time. The power, if not effective, is stored, and no piece of ore, not crushed, remains between the rolls; it falls by gravity, and is returned to the roll, if too large, by an elevator. No cushioning is possible. Stamps have a maximum of velocity which is very soon reached, beyond which the fall is reduced. It may be carried so far that the stamp head may be kept suspended in the air, and consequently do no duty whatever; while the increased speed of rolls always gives increased output, although beyond a certain limit the power may not be so economically expended, but there is no cessation of action possible as in the stamp. If we make a comparison of parts required in the construction of each machine, we shall find that the two machines of the same capacity will have the following working parts. For the rolls:

4 journals on two roller shafts.
4 pillow-blocks for journals.
2 crushing rollers.
2 side plates.
12 wearing parts for one set of rolls.
Or 24 parts for two sets of rolls.

All these parts are made so strong that they are not liable to break, and the wear is not rapid. They can all be easily replaced, and all are easily accessible at any time. A 50-stamp mill has

15 journals.
5 cam-shafts.
15 bearing boxes for journals.
50 stems.
100 guide boxes for stems.
50 stamp heads.
50 shoes.
50 dies.
50 cams with keys.
50 tappets.
Or 435 working parts.

All of these parts are subjected to rapid wear, many of them are liable to break, and it requires much time to watch them, for if left to take care of themselves a break or a jam would cause much damage. Most of them are accessible to make repairs, one or more of the stamps, or even batteries, may be hung up to

make them, and while any single repair does not consume much time, the multiplicity of them in the course of a year does. The screens have not been considered, as they are common to both machines, although as they receive part of the blow of the stamp which thrusts the crushed ore against them with considerable force, they must wear more rapidly in the stamp mill than in the roll where the ore flows on them only by gravity. This simple comparison of the parts speaks for itself.

It has been found by experience that the condition of the crushed ore from rolls makes the pulp much better suited for lixiviation, which seems to be the process of the future, than that from the stamps, as the particles of ore coming from the rolls are not only more uniform in size, but [illegible] in much less dust, which interferes with the rapid and effe[illegible] passage of the solutions. It has been found, too, that in c[illegible]orurising roasting in mechanical furnaces previous to amalgamation, great fineness of the ore is unnecessary, except in the rare case when the precious metal is very fine and is evenly distributed through the gangue, so that, with the exception of this single case, rolls seem to give the most favourable results.

To compare the wear and tear of the stamp mill and of the rolls running at full capacity, it may be assumed that they are in each a constant quantity for any twenty-four hours, as is shown by the experience of the following mills working on very different ores, and having a very different capacity.

	Kind of Ore.	Number of Screen.	Weight of Stamp.	Number of Drops per Minute.
Manhattan Mill, Nevada ...	hard	50	1000	100
Ontario Mill, Utah... ...	soft	30	850	92

In these two mills the wear and tear for actual horse-power expended was very nearly the same over a period of several months.

The wear of rolls is almost exclusively confined to the steel tyres and cheek pieces, that of the battery to the great number of parts enumerated above. Something over 80 per cent. of the steel of the rolls can be safely used in crushing before it is necessary to put on new tyres. Generally it is not safe to wear off more than 50 to 60 per cent. of the shoes and dies before replacing them.

On account of the number and complicated nature of the parts of the battery, skilled labour has to be used to a considerable extent, while only a nominal amount of it has to be used with the rolls. Every one who has ever seen a mill, knows how often the stamps have to be hung up, while rolls, if they are made strong enough at the outset, require little or no repair except the change of tyres, which is quickly done and at long intervals.

The statistics of the wear of rolls is as yet confined to the experience of the Bertrand Mill and Mount Cory Mill, and while it is not generally safe to draw conclusions from very small data, the comparison in this case may be safely made if all the circumstances are taken into consideration. The rolls were introduced in the Bertrand Mill on the idea that the ore should be made to pass a 30-mesh screen; when they found it advisable to crush coarser, no appreciable difference was noticed in the output of the rolls, because the screening and elevating capacity was not increased. This had been calculated for only 60 tons in twenty-four hours. When the screen and elevator capacity was increased, the output went up at once to 50 tons in twelve hours, it being still limited by the screens and elevators. This has led to the practice of using the rolls in the daytime only, since they have now reached the capacity of the leaching plant. This of itself is a great boon to the men, as it makes them work day shift only. From this it will be seen that we do not actually know what the capacity of the rolls is, that, in any case, as their output has been limited by the capacity of the screens, the capacity so far attained will be a minimum. At the Bertrand Mill two sets of rolls crushed 20,000 tons of ore through screens varying from 30 to 16. These same sized rolls have a capacity of 100 tons in twenty-four hours crushed to pass a 30-mesh, or 150 tons to pass a 16-mesh screen. Estimating the actual wear as 250 days, the cost of the wear will be:

Two sets of steel tyres at New York	$524.00
Freight on 3264 lb. at 3 cents	98.00
Total wear	$622.00

The wear for twenty-four hours' work will be:

For steel tyres	$2.48
Other wear, screens, supplies and lubricants (maximum)	1.75
Wages for repairs (maximum)	1.25
	$5.48

It is not yet possible to make an accurate comparison of the cost of working by stamps and rolls. The stamp mill has been used so long, that there is plenty of data for it, but rolls have been so recently introduced for metallurgical purposes, and have been used on a large scale in so few mills, that sufficiently accurate data, for all the cases to be considered, has not as yet been accumulated. For the purpose of a comparison of expense we will say that two sets of 26-in. rolls are equal in capacity to 30 stamps instead of 50, as we have above shown, with stamps weighing 850 lb. to 900 lb., dropping 90 to 95 drops a minute, with a fall of from 7 in. to 9 in. It has been found at the Bertrand Mill that two sets of 26-in. rolls, with the consumption of four cords of wood, will crush 100 tons of moderately hard quartz ore in twenty-four hours so that it will pass a No. 16 screen. A 30-stamp mill to do the same work will consume six cords of wood. We will suppose that the conditions of labour and prices of material at both mills are the same. An ordinary stamp battery,* including hardwood screens, frames, guides, pulleys on the cam shafts, Tullock's feeders with iron hoppers and necessary bolts, weighs about 90,600 lb., and costs in Chicago $5850. The frame takes about 36,000 feet of timber. The expense of the stamp battery, not including elevators, conveyers and revolving screens, which are common to both, will be:

Plant at the foundry	$5,850.00
Freight to locality	2,718.00
Lumber	1,800.00
Cost of setting up	4,000.00
	$14,368.00
Cost of building in excess over a building for rolls	1,500.00
Cost of engine boilers in excess over rolls	1,250.00
	$17,118.00

The amount of lumber required for the rolls is nominal. As the machine has but few parts, the labour of setting it up will be nominal. The weight of one set of 26-in. rolls is 14,700 lb., and the cost in New York is $2250. They require one self-feeder weighing about 1000 lb., say, costing $200; therefore,

* "Production of Gold and Silver in the United States." Washington, 1883, p. 740.

The cost of two sets of 26-in. rolls and one automatic feeder	$4,700
Freight	912
Cost of setting up, including lumber...	700
Total	$6,312
Difference in favour of rolls	$10,806

For the wear of the stamp the data has been taken from the Manhattan and Ontario Mills, whose statistics have been given above, and the Lexington during the first year's run. Making a little allowance for the extra breakage from the heavy stamps at the Manhattan, and the absence of it from the newness of the Lexington, the following estimate, which appears impartial, but the correctness of which time only can show, has been made.*

Wear of a 30-Stamp Battery for Twenty-four Hours' Running Time.

Wear, breakages, supplies, screens, and lubricants ...	$11.50
Wages for repairs	5.50
Total cost	$17.00

The cost is distributed as follows:

Wear of shoes and dies	$40
Tappets, bosses, cams, stems, shafts, flanges, and boxes	38
Screens, screen frames, battery guide, lubricants, carpenter and machine supplies	22
	$100

The wear and tear of rolls has been found to be $6.45. The difference, therefore, in favour of them, is $10.55. Taking now into consideration the interest and sinking fund, which should not be neglected considering the precarious existence of the mines, and should not be put, for that reason, at less than 15 per cent., if we suppose that the mill works 350 days in the year, and consider that the mill will cost $10,938 more than the roll, we find that the interest for the sinking fund account will be $4.68 per day.

Saving from the use of 26-in. Rolls as compared with 30 Stamps.

Wear and tear and repairs	$11.52
Interest and sinking fund	4.68
Two cords of wood at $6	12.00
Total saving	$28.20

The saving will probably be somewhat more than this, as it has been previously shown that the crushing surface of the 26-in.

* "Production of Gold and Silver in the United States." Washington, 1883, p. 742.

rolls was equal to that of a 50-stamp mill; admitting, however, that it was only half that amount, the saving is still sufficient to merit the serious attention both of mill men and engineers. If nothing more than the freedom from dust and noise was to be gained and the work was not quite so well done, there would still be sufficient in favour of the roll to make it worth while to use it, were it only for the compactness of the machine and the freedom from stoppages.

There is an additional argument in favour of the rolls, which is that they can be sent out from the machine shop ready to be set up, and as they are self-contained can be put up ready for work in a very short time; whereas the number of parts to a stamp battery are such that it must be sent out in pieces, and consequently requires months to set it up, supposing that its complicated foundations are all ready to receive it.

One great difficulty in the way of introducing rolls as a substitute for stamps is that when people decide from motives of economy to adopt rolls, they want something very cheap, which leads them, for the sake of saving a little at the start, into great expense in the end by purchasing a really poor machine. Nothing is more costly than a cheap machine, when, as in this case, the economical result can be obtained only by great strength and superior construction. Even the Krom rolls, which are the strongest and best constructed of all the varieties now in actual use, might have their weights, in certain parts, advantageously increased. This remark applies equally well to rolls for lamination as well as for crushing. Dead weight here is an element of strength and not of weakness, as it is in some engineering structures. Whatever else is true, those rolls which are to be successfully used in ore dressing works must have great strength, simplicity of design, all the wearing parts easily accessible, and so constructed that all repairs can be easily and quickly made, for they are to be used where skilled labour is dear, and repairs both difficult and expensive, and as stoppages mean not only outlay for repairs but cessation of output, and liability to them means diminution of capacity, they should be provided against in every possible way. From a mechanical point of view it is remarkable that rolls were not perfected many years earlier.

CHAPTER V.

ROASTING SILVER ORES.

Most of the ores which are treated in the United States require to be roasted. A few of the ores which are known as free milling ores, consisting of either the metal or easily decomposed compounds of it, do not require roasting, and they are crushed wet; but the rebellious ores require to be roasted, for which reason dry crushing mills are generally used for them. Where ores which must be roasted are crushed wet, they must be first dried, which involves the use of drying floors, and somewhat increases the size of the plant. In the early days, heap roasting was used exclusively, and is still when the pieces of ore are of large size, and it is not desirable to crush them. Stall roasting has been used, but does not appear to have been very successful. The only furnaces which were at first successful were the reverberatory furnaces. This furnace was imported from Europe, and was constructed in several different ways; the different varieties of them are described under each separate case. They were either single or double-hearthed, in the latter case the hearths being arranged one above another, as in the case of some of the gold-roasting furnaces in Grass Valley; or one in front of the other, as is the case with some of the gold-roasting furnaces in Amador County, California; or they were step furnaces, as at the Lincoln Mine, in California, and of the Boston and Colorado Works, in Colorado.

Silver ores were formerly generally roasted in reverberatory furnaces with a single hearth, the salt being added so as to form chlorides, the object being first to drive off the sulphur and then to transform the metal into a chloride. When, however it became necessary that the works should have a large output, it became apparent that reverberatory furnaces had a very limited capacity, unless the space available for the erection of the works was very large, and the capital which

could be invested was not limited as it usually was in the starting of the first works in the West. It was also found that the success of the roasting in the reverberatory furnace depended entirely on the care and skill of the men, great skill being required where accurate roasting is done. In a country like the West, where every man expects to make a fortune, it was found impossible to keep the men long enough to acquire this skill; and when complaints were made of the way the working was done, the workman simply transported himself to another locality, and this soon became a great inconvenience. In European works, where the population is more stable, a system of fines can be used as at Przbram,* to insure the careful roasting of the ore; but in the United States such a system would be entirely impracticable. The works here were therefore entirely at the mercy of the men, who, to do them justice, did their work with moderate care, but, when it was not successful, did not feel themselves responsible, and if they were blamed, simply left the works with the certainty of finding profitable employment elsewhere. The small output which these circumstances necessitated, led very soon to the invention of furnaces for doing away with skilled labour and increasing the output, which, except in special cases, have driven out the reverberatory furnace. A large number of these furnaces have been invented under the name of mechanical roasters. Most of the large mills now employ them exclusively, and the time seems to be not far distant when their use will be universal. These furnaces have the advantage that they require little or no skill on the part of the workmen, that the whole operation is under control from beginning to end, and the whole work of roasting the ore, both introducing it into the furnace, roasting and discharging it, and carrying it off to the cooling floors, is almost entirely automatic. There are but few types of these furnaces. Each claims, according to its success in a particular case and the failure at others, which often does not depend on the furnace or the process, but upon incidental circumstances, a higher percentage of useful effect than the other. Many also base their superiority on the low or lower first cos of their particular kind of structure. It would not be profitable

* "School of Mines Quarterly," vol. iv., p. 128.

to discuss each separate type, the general principles remaining the same.

It may be said that almost any roasting furnace that does not fail in its mechanical details, and the different parts of which are of such light weight that they can be easily transported into the districts where they are to be built, provided they are adapted to the fuel which can be obtained in the country, will, if used with ordinary intelligence, be successful. In the early days of these appliances dust chambers were not used, and a large quantity of material was lost. When the dust chambers were introduced the material that was collected in them was simply taken out and re-treated as original ore. The improvements recently made have consisted in introducing auxiliary fires, so that the material once charged into the furnace has not to be handled except as roasted material, the auxiliary fire doing the work of roasting on the ore which is carried over by the draught, so that the material from the dust chambers, when it is collected, is as completely roasted as that taken from the body of the furnace. In the early days great stress was laid upon having the interval between the roasting of the ore and its treatment as short as possible. It has been found, however, by recent experience that this is a mistake, and that a very considerable percentage of chloruration takes place in the ore after it has left the furnace, provided it was left in heaps; so that the practice now is to have a decided interval between the two, during which time the excess of salt contained in the ore acts with the greatest efficiency on it, so that the chloruration is raised more than the value of the time lost, by allowing the material to remain in a heap.

These mechanical furnaces are of two general types, revolving cylinders and shaft furnaces. Of the former there are many varieties; of the latter only one, which bears the name of the inventor and is called the Stetefeldt furnace.

Brückner's Cylinder.

Of the revolving cylinder furnaces, the Brückner furnace was invented by Mr. William Brückner, and erected in San Francisco in 1864, was introduced into Colorado in 1867, and since then

has been erected and successfully used in most of the silver and gold-producing States and Territories of the West. It was introduced for roasting gold ores, and rendered the extraction of 90 per cent. of the gold possible, but it is now almost exclusively used for silver ores, and has rendered a real service in the working of that metal in the West. There are many of them in the Territories, which were used in the extraction of nearly one-half of the silver produced there in the years 1855-6. The idea of this furnace was suggested to Mr. Brückner by experimenting with two truncated cones to make the ore fall from the ends of each of them into the other and return, and so secure a constant agitation of the ore by mechanical means. The cylinder may be regarded as two such cones, and the diaphragm as the points of intersection at different intervals of their revolution.

It consists of an exterior cylinder of boiler iron 12 ft. long and 5 ft. 6 in. in diameter, the ends of which are closed, leaving an opening in the centre of each, 2 ft. in diameter. This opening has a flange which projects several inches on the outside. One of these openings connects with a fireplace, and the other with a flue leading to the dust chambers. In the first furnaces constructed the cylinder was closed with a head at right angles to it, having the flange fitted on at right angles to the head. The ends of the cylinder are now made conical, which simplifies the construction of the interior of the furnace. The method of construction is shown in Figs. 96-100. About the middle of the cylinder there is an opening for the introduction of the charge, which is closed by an iron door. Two bands with square projections are bolted on the outside, near the ends, each one of which turns on two friction rollers which support the cylinder. The one near the flue fits into the wheels, which are provided with flanges for the purpose, and prevent any tendency that the cylinder may have to slip out of place. The one near the fireplace simply runs on the friction rollers. Between these two bands, and nearest the flue, is a circle of gearing, which is cast in one piece and carefully turned, so as to secure an even revolution. It fits into a spurwheel, which gives the motion to the furnace. The gearing should be compound, so as to allow of two speeds

BRÜCKNER'S CYLINDERS FOR ROASTING SILVER ORES.

Fig. 96.

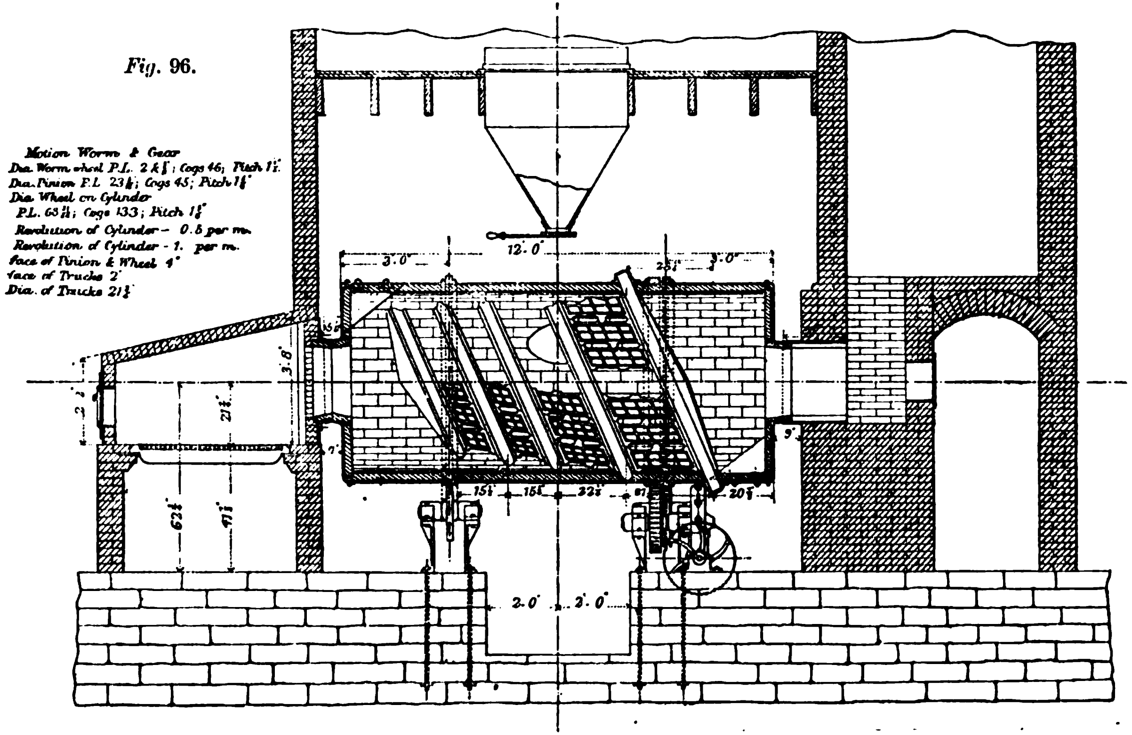
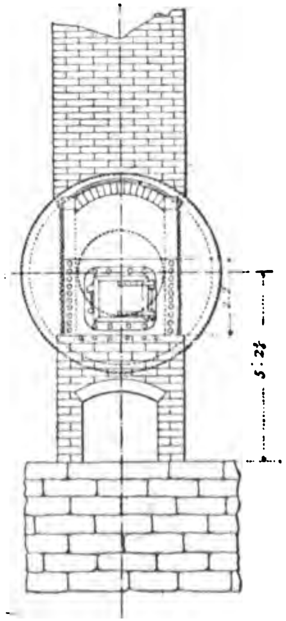


Fig. 97.



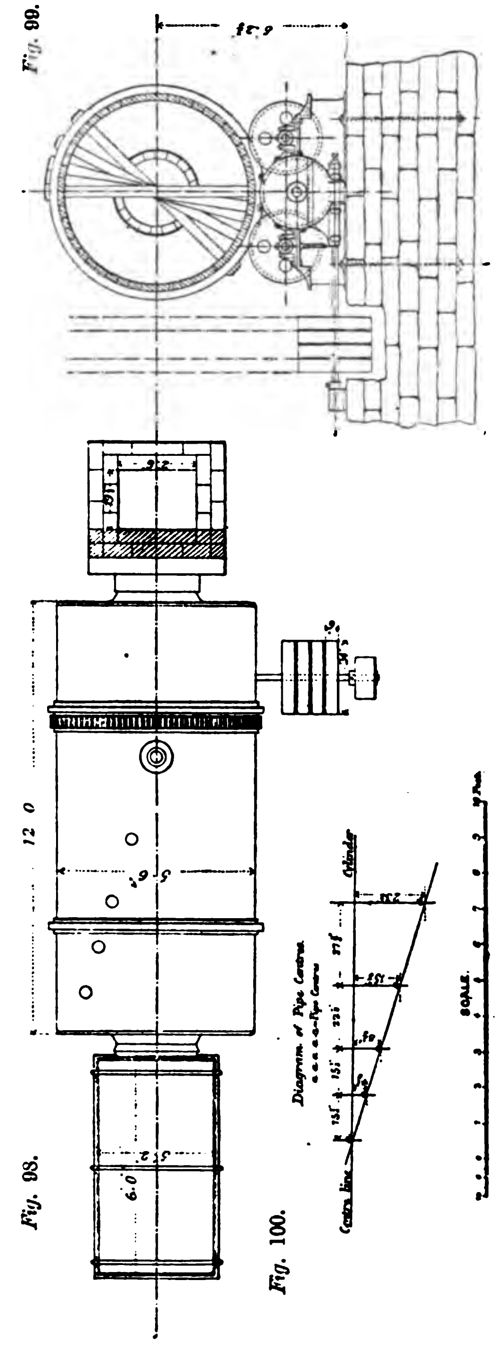


Fig. 99.

Fig. 98.

Fig. 100.

which are required at different stages of the process, and should be so arranged that where there are a number of cylinders, any one of them may be stopped at pleasure without interfering with the others. In order to provide against the possibility of settling, each journal box of the friction rollers is held in position by adjustable screws, so that it can be moved laterally or perpendicularly.

Passing through the cylinder from side to side are six pipes, which make a diaphragm in the form of a grate. They are inclined at an angle of 15 deg. to the axis of rotation, making at the same time an angle of 30 deg. to 35 deg. to the plane of this axis, as is shown in Fig. 99. The tubes of the diaphragm pass through to the outside, so that air constantly circulates through them. It was expected that the cooling of the air and the formation of a scale would protect them from the action of the sulphurous vapours; this has proved not to be the case. The object of the diaphragm is to force the charge to continually move backward and forward from one end of the furnace to the other; to raise it as the furnace revolves and allow it to fall down through the interstices of the bars, so as to expose the largest amount of the ore to the action of the fire. The whole interior of the cylinder is lined with one layer of ordinary red brick laid flat and set in mortar made of one part fireclay and two parts firebrick thoroughly mixed and beaten together. At the Pelican Mill the lining is anchored by means of irons bolted for that purpose to the iron casing of the cylinder. The brick is cut upon one side in order to form a complete arch in the interior of the furnace. At the Niederland Mill the brick is put in without shaping, and each half cylinder wedged from the diaphragm, so that no anchorage is necessary. The neck bricks are moulded for the purpose. The time that the lining will last depends upon the care with which it is put in. If well laid in good fireclay mortar it will generally last a year and a half without loosening; it usually becomes quickly coated with a crust of sintered ore, which protects it from abrasion.

In the early construction of the furnace the ends were closed by rectangular pieces, and the lining was made conical to reduce it to the proper size. This was found to complicate the construction and necessitated frequent repairs. The ends of the cylinder

are now made conical and the lining made of the same thickness throughout. The weight of the cylinder is thus considerably reduced, and repairs to the lining are much less frequently necessary. In the furnaces first built the cylinder was set on a foundation of masonry and the rollers supported on timbers. This construction caused so much trouble that it is now supported on a cast-iron frame which is carefully adjusted before it leaves the shop, thus greatly simplifying the erection of the furnace at the works. The projecting flanges fit loosely into the firebox at one end and into the flue at the other. Over the lower part of the flue end a piece of sheet iron is placed inclined so as to throw any ore which might tend to escape through the opening between the cylinder and the flue, back into the furnace. Exactly opposite to the opening, a door is placed in the flue so that the working of the furnace may be examined at any time. The fireplace may be built entirely of masonry, or the sides may be made of boiler plate tied with rods and lined with brick, the roof being arched with brick without any ironwork above it except the tie-rods. It is usually 6 ft. long and 3 ft. 2 in. wide. The height from the grate to the roof at the door is 2 ft. 2 in., and at the neck of the furnace 3 ft. 8 in. It generally lasts from six to eight months. The outside iron boxes have lasted two years, but have sometimes been burned out through carelessness in a shorter time. A circular opening is made in the back part 6 in. above the grate, to admit of the entry of the neck of the cylinder.

The throat of the furnace is lined with firebrick. Each cylinder has its own dust chamber, which is cleaned on Sunday. At the Niederland Mill the fine dust goes into a flue 40 ft. long, 6 ft. high, and 7 ft. wide. The coarse dust falls into a receptacle made for that purpose near the furnace, and is drawn out in a box below. The flue passes under the drying floor and furnishes part of the heat used there. The amount of fine dust caught is 10 tons in the dust chamber, and 10 tons in the drying kiln flue per month. In addition to this 5 tons of the coarser variety are taken from the boxes at the mouth of the flue. The coarse particles are put back at once into the furnace. The dust contains on an average 32 oz. of silver. When there is enough collected for a charge it is treated by itself with a little ore and salt. At the Pelican Mill

the amount is 1500 lb. for each pair of cylinders per week for the heavy ores. With light ores it is 10 per cent. less. Sometimes a steam jet is introduced into these chambers, with the object of moistening the fine dust, and causing it to fall.

The line shaft which runs the furnace should make about twenty-four revolutions per minute. It takes about three horse-power to drive one furnace. As much of the furnace as is possible should be made of wrought iron, as castings are not so strong and much heavier, and when it is desirable to erect furnaces in regions which are not very accessible they increase the expense. The total weight of all the ironwork is 1600 lb. It is all made at Chicago or Cincinnati, and sent out to the works.

Any kind of ore may be treated in the furnace, but the higher the percentage of sulphur and galena the smaller the quantity that can be roasted in twenty-four hours. Many of the ores of Colorado are very refractory, containing large quantities of lead, zinc, and sulphur. They are very difficult to treat owing to the tendency which they have to form either fusible compounds, to clinker, or at least to cake, and thus form masses which are not affected by the salt, and must be re-treated. The greater the amount of sulphur in the ores the longer the time it takes to treat them. The difficulty is greatly increased with the tendency of the ore to cake. All ores must be crushed fine before they are charged.

As soon as the previous charge has been withdrawn from the furnace it is ready for a fresh charge. It is at a dull red heat from the previous charge, or is brought into that condition revolving at the rate of one-half to one turn a minute. It is then brought into position with the charging door up, and stopped. The ore, which is stored in bins in the story above, is charged from a hopper through a long flexible conduit, which is brought directly over the charging hole and the charge introduced. The weight of the charge is very variable, and depends upon the nature of the ore. At the Niederland Mill, in Carabou, where the ore contains 5 per cent. of galena, 4 per cent. of blende, and 2 per cent. of copper pyrites, or a total of only 11 per cent. mineral matter, the charge is 3700 lb. as a maximum. At the Pelican Mill, where the ore contains 15 to 16 per cent. of galena and pyrites, and sometimes as high as 15 per cent. of blende, or 30 per

cent. of mineral matter, the charge is rarely higher than 3500 lb., and sometimes considerably less. These ores are very difficult to treat on account of the very large quantity of blende. The size of the charge is, however, not necessarily an indication of the capacity of the cylinder, for the time taken to treat the ore is exceedingly variable, depending upon the care that must be taken with it. The greater the amount of mineral matter the larger the amount of sulphur will be, and the longer the time it will take to treat it. If the ore is very "light," that is composed mostly of oxides, the charge may be large and the time as short as four hours. At the Niederland Mill it takes eight hours. If it is very "heavy," like those of the Pelican Mill, it will be at least twelve to thirteen and sometimes twenty hours. The four cylinders at the Niederland Mill roasted, in the years 1875-6, nearly 4000 tons of silver ores. The capacity of the cylinder for each variety of ore is determined by filling the cylinder, so that when the ore has swelled to its maximum it will just run out of the back nozzle of the cylinder.

As soon as the charge is introduced a sliding valve in the bottom of the hopper cuts off the ore. The cylinder door is then closed and fastened, and it is made to revolve one turn in two minutes. For heavy ores, that is for ores which contain a large amount of sulphur, the cylinder at the time of charging must be very hot, in order to get the sulphur burning as soon as possible, which generally takes about an hour. When the ores are light, that is, consist mainly of oxidized products, they are simply heated and then chlorurised. Sometimes, as in the Niederland Mill, the salt is charged with the ore. The sulphur is allowed to burn as long as it will, with as much air as possible. The fire is not made active until the sulphur will no longer burn. The damper for each cylinder is shut down when the ore is introduced, and is kept so until the sulphur begins to burn, when it is raised. With heavy ores the sulphur burns from three to five hours, during which time the fire is only just kept alive on the grate. It requires from the time the sulphur ceases to burn from five to six hours to complete the roasting, during this time the furnace is gradually raised to a red heat. It is sometimes found advantageous to regulate the fire by having water in the ashpit.

During all this time the diaphragm causes the charge to move backwards and forwards. So long as there is any sulphur in the ore it falls freely through the diaphragm and around the furnace. When the sulphur commences to burn out, it does not fall continuously, but begins to break as it falls.

From ten to twelve hours from the time it is charged it is ready for salt, which is introduced through a hopper. For a 3500 lb. charge not less than 200 lb. or more than 250 lb. of salt are required, depending on the richness of the ore. Soon after the salt is introduced, the ore becomes spongy from the double decomposition of the sulphates formed during the previous roasting chlorine being given off. When it is chlorurised, there is no smell of sulphurous acid. There must be a clean smell of chlorine given off for about half an hour before the charge is done. Samples are taken from the door in the back of the flue, or with a spoon on a long iron rod, through the door of the fireplace, and sometimes by opening the door of the cylinder, and allowing a certain quantity to drop into the car as it revolves. The chloruration varies from 85 to 95 per cent., according to the ore, and the care with which it is treated. When the same ore is treated it is not always assayed. In some works the workman is allowed to judge by the eye and the smell as to whether the chloruration is properly done, which is a very bad practice. It should always be assayed with hyposulphite of soda at different stages of the process.

When the charge is finished, which is generally in from four to thirteen hours after the charge is introduced, an iron wagon is run underneath the cylinder and the charging door removed. The cylinder is allowed to revolve, with the fastest motion, with the door open. The charge falls into the wagon and is carried to the brick cooling floor. At the Pelican Mill this wagon is 5 ft. 6 in. long, 34 in. wide at the top, and 29 in. at the bottom. In some works the ore is dropped into a hopper beneath the furnace, at the bottom of which there is a screw or endless chain which carries the ore out into an iron trough cooled with water. This avoids a considerable waste of time in cooling the ore and some labour. It takes from one to one and a half hours to discharge the cylinder. The chloruration is alway higher when the ore is left for some time in a heap before it is cooled.

Before chloruration the Colorado ores are greyish, and after the chloruration they are a brownish red. The whole art of chloruration consists in putting in the salt at the proper time, while there is still some sulphur in the ore. It is then said to have a velvety look and must be entirely free from lumps. The temperature should never be so high as to sinter the ore. This never happens except with green hands. It is impossible to prevent some of the ore from caking and becoming attached to the sides of the cylinder. This is scraped off and must be crushed and re-treated, for which purpose the cylinder is charged with 3000 lb., 500 lb. of which is raw ore, and with 160 lb. of salt. The ore is always screened on the cooling floor before amalgamating it. The quantity of screenings is such that one of the five cylinders will be run upon them two days in a week. The exact quantity will depend largely upon how long the hot ore remains in the wagons, and how long it remains in heaps on the cooling floor before it is spread out. All heavy ores have a tendency to cake in the heaps if they remain for any considerable length of time. The scrapings of the five cylinders, at the end of a week, when working on heavy ores, will amount to one charge. The quantity of scrapings depends on the amount of lead in the ore. With light ores the cylinders have been run two weeks without any scrapings. The scrapings and screenings are crushed together and are always treated separately from the ore.

One and a half cords of wood is more than enough at the Pelican Mill to run two cylinders twenty-four hours with very heavy ores. This is three-fourths of a cord of fuel for 3500 lb. At the Niederland Mill they use one and one-half cords to 5 tons of ore, which is still less, being three-tenths of a cord, but the ores are quite light. Mr. Brückner states that three-quarters of a cord of wood, or three-eighths of a ton of coal, must usually be counted on for ordinary ores. The wood is piled beside the cylinders at the workman's hand. The two cooling floor men bring it to the cylinders. The woods which are generally used are red spruce, which costs $5 per cord, and bastard pine, which costs $4. The red wood gives a very quick fire, and a great amount of flame, which is important for the proper working of the cylinders. Five furnaces require one roaster and one helper for a shift of

twelve hours, or four men in twenty-four hours. It is very questionable whether it is desirable to have such long shifts in a process which requires such constant watching. With eight-hour shifts the men would be less fatigued and much more likely to do the work well. The cost of roasting at the rate of 20 tons a day with four cylinders at the Carabou Mill in 1871 was:

Two roasters	$200.00
One helper	75.00
104 cords wood at $3.50 per cord...	364.00
26 tons of salt	1820.00
Oil	2.50
Candles	5.50
Tallow	1.50
Black lead	1.00
One-third power and general expenses	287.00
Cost of roasting 520 tons	$2756.50
,, ,, 1 ton	5.03

This is very much less than roasting with a reverberatory furnace. The expenses for roasting light ores in Georgetown with two cylinders, having an average capacity of 7 tons in 24 hours, were:

One man for two cylinders 12 hours at $3.25 ...	$6.50
7 per ct. of salt, or 980 lb., at 3 cents per pound	29.40
1½ cords of wood at $5 per cord	7.50
Total for roasting 7 tons of ore	$43.40
or for 1 ton of ore	6.20

The expenses for labour and fuel are small, but vary somewhat in different localities. The roasting is very uniformly done, occupies less time than in a reverberatory furnace, and costs less.

The only repairs required are to the throat and the diaphragm. The throat must generally be repaired once in six weeks or two months. In the Territories the castings are usually made from old iron taken from all kinds of machinery and furnaces, which has been frequently melted, without much regard to quality, and is very hard and poor. They cost 8 cents per pound. They are so bad that the castings are now always sent from the East. The time that a diaphragm will last depends upon the quantity of sulphur in the ore. It will usually last from four to five months with very heavy ore. With light ores one set will last a year. When a tube of the diaphragm breaks, the cylinder is still

run for the week, and new tubes are put in on Sunday, when the works stop for repairs. A great deal of importance was at first attached to having the diaphragm in good order. It was found that the scale which should protect it would not always form, and the tubes were constantly giving out and being replaced by new ones. At the Niederland Mill there had never been, since it was first erected, a full set of diaphragms in the cylinders; new diaphragms complete were put in, but they were rapidly worn out, and as they broke the stumps were in the way, and were taken out and not replaced. It is now found that the furnace works better without them, and since the use of diaphragms has been definitely abandoned, it is found that the furnace works just as well, and there is a great deal less dust in the dust chambers, and with ores containing lead, fewer lumps which have to be crushed and re-roasted. The rotary motion of the cylinder appears to be quite ample to insure sufficient movement in the ore to have it thoroughly oxidized. In the recent constructions of this furnace,* two and sometimes three cylinders, placed at different levels, discharging the one into the other, have been used with great advantage. The operation may be a continuous one from one furnace to the other, or separate charges may be treated.

The cost of one cylinder complete, including all the machinery and ironwork, with the royalty, is about $2500 in Cincinnati, so that a single furnace delivered in Colorado will not cost less than $3000 to $3500, depending on the accessibility of the district where it is to be erected. The royalty on the cylinders has been reduced several times. In the year 1874 it was $1000.

The cost of a double-cylinder plant erected in Montana is given below.

IRONWORK :		
2 cylinders, 20 ft. by 7 ft.; 2 fireboxes; 1 smoke stack, 60 ft. by 30 in.; 2 conveyors; 1 hopper and driving gear; driving shaft, pulleys, pillow-blocks, &c., complete		$5,850.00
MATERIALS :		
20 M firebricks, at $22.50	$450.00	
16 bbls. fireclay, lime, and cement ...	256.00	
		$706.00

* Trans. Am. Inst. Min. Eng., vol. xiv.

Brought forward ...		$6,556.00
Labour:		
1 machinist and 2 bricklayers, 20 days ...	$360.00	
3 helpers, 20 days	180.00	
		540.00
Building:		
Frame building, 60 ft. by 20 ft.		2,300.00
Freight:		
40 tons from Chicago, at 2 cents per pound... ...		1,600.00
Total cost		$10,996.00

The cost of roasting from 20 to 40 tons a day of refractory ore with such a furnace is estimated as

2 roasters, at $4.00	$8.00
2 cords of wood, at $5.00	10.00
Oil lights, and extras	2.00
	$20.00

At the Anaconda Works, in Montana,* the cost of roasting 8½ tons of concentrated ore in twenty-four hours, containing 37 per cent. to 40 per cent of sulphur, is:

2 roasters, at $3.50	$7.00
2¾ cords of wood	11.00
Oil lights, &c.	.20
	$18.20

Cost per ton, not including the cost of power, $2.14.

By arranging six cylinders as two sets of three furnaces instead of three sets of two, it is estimated that the cost would be:

2 roasters	$7.00
5½ cords of wood...	22.00
Oil lights, &c.	.60
	$29.60

For exactly the same ore and a daily capacity of 51 tons, the cost per ton is only $0.50.

When the furnace was first introduced, insufficient experiments were made, and, like most good things, more was claimed for it than could be accomplished. This put a check on the introduction of the cylinder for a short time only. The advantage of the cylinder is that it does its work well and uniformly, and that the ore is always under full control of the workmen; that it uses a small quantity of fuel and labour; that

* "Engineering and Mining Journal," vol. xli., p. 166.

the percentage of chloruration is high, and may be carried to 96 or 97 per cent., if sufficient care is taken; that it does not require special labour, as the process is easily learned by any one; the men are usually anxious to learn it, as they consider the position a responsible one, and that the machinery is simple, not likely to get out of order, and easily repaired when it is damaged. It is especially useful in mills of small capacity.

The Brückner cylinder was successful from the start, because the men having no heavy work to do could give their entire attention to the management of the fire, and consequently had leisure to watch the condition of the ore more carefully than they would have done if they had been obliged to do the manual work at the same time. It is true that the condition of the ore in the reverberatory furnace could be felt with the tools. It is also true, however, that the work is not likely to be as well done where the men are obliged to do heavy manual work and exercise great judgment at the same time. The chloruration tests in the Brückner cylinders, where they were managed with any judgment at all, are generally higher than those from the reverberatory.

The success of the Brückner cylinders made many imitators, most of whom, either with a desire to have something original, or to make alterations which were not always improvements, have changed either the shape of the furnace very slightly, its method of rotation, the shape of the interior bricks, length of the furnace, and its inclination. As soon as the success of the auxiliary fireplace in the Stetefeldt furnace became known, some of these added the right to use this fireplace, and have gained great reputation for their furnaces by so doing. It became evident, shortly after the Brückner cylinder was introduced, that the diaphragm could not be maintained. The furnace could not stop after the diaphragm became damaged or fell out altogether, and, in either case, the furnace was found to work just as well without it as with it. When it was abandoned a large number of imitations of it without the diaphragm were constructed, either to avoid the royalty or to get a new patent.

The Brunton Furnace.

In this furnace the shape of the outside is oval; the diaphragm is omitted. The furnace is rotated by friction instead

of by gear. The shape of the furnace is well adapted to keep the bricks in, so that there will be less loss by deterioration of the brickwork. Furnaces 12 ft. long by 6 ft. in diameter have been used. Such a furnace, including the bed-plates for the rollers, and their gearing, would weigh about 10 tons.

The Pacific Furnace.

This furnace is a sheet-iron cylinder of large capacity, lined with brick, and differing in no way from the Brückner, except that it is capable of treating a large quantity of ore in twenty-four hours, 7 tons to 8 tons.

The White Furnace.

This furnace is cylindrical, constructed of cast iron made up of short segments bolted together; a brick lining is placed in these segments, and contains projections which cause the ore to change its place as the cylinder rotates. The cylinder has an inclination either to or from the chimney arranged in such a way that it can be adjusted at will. The ore is charged either at the flue or at the fireplace according to circumstances. When charged at the fireplace the results were entirely unsatisfactory. When charged at the flue the lighter material is carried directly into a series of dust chambers, so that it is necessary to have the auxiliary fire of the Stetefeldt furnace to roast the dust which passes into it. The portion not carried off by the draught descends towards the fireplace, so that it becomes heated progressively and is discharged into a pit at the end of the furnace between it and the fireplace, where it is exposed for a considerable time to the flames passing over it, thus providing for the chloruration after the material leaves the furnace.

The Howell Furnace.

This furnace consists of a cylinder which is lined with brick nearly one-third of its length, the other two-thirds of the cylinder towards the fireplace being cast iron only. In order to have a uniformity of size the part of the furnace towards the fireplace is the diameter of the thickness of the bricks greater than the flue end. The inclined portion of the cylinder is filled with iron projections, which keep the ore constantly changing its position before it arrives at the hottest part of the furnace. The

cylinder is 23 ft. 2 in. long, the diameter of the iron shell varies from 22 in. to 50 in., and of the brick-lined part from 32 in. to 60 in. The output of the furnace is determined by the number of rotations permitted, which are adjustable, and also by the inclination. The quantity treated varies from 8 tons to 45 tons, depending on the character of the ore and the size of the furnace. A 60-in. furnace requires from six to eight horse-power to run it. The first Howell furnace was built at the Citizen's Mill in Austin, Nevada, in 1872. Abroad it has been called the Oxland furnace. The preliminary results were not in any respect more favourable than with any other roasting furnace, but since the addition of the auxiliary fireplace the working has been excellent.

Below are given some of the results of roasting with this furnace.

AVERAGE for Seven Days from June 17th to 23rd.

—	I.	II.	III.	IV.
Solubility	18	30	34	30
Value after leaching . .	$140.16	$175.45	$183.84	$180.45
Value of ore . $161.10				
I. Roasted ore from cylinder . . $114.47				
II. Roasted ore from flue below auxiliary fire $122.55				
III. Roasted ore from dust chamber . $120.66				
IV. Roasted ore from chamber outside . $126.32				

The following Table gives the chlorurations of roasted ore from the Howell furnace and from the flue and dust chambers directly after drawing:

	Chlorurations Roasted Ore from Cylinder.	Chlorurations Roasted Ore from Flue and Chambers.
	Per cent.	Per cent.
June 8	47	65
,, 9	59	67
,, 10	52	77
,, 11	67	76
,, 12	70	70
,, 13	73	87
,, 14	52	83
,, 15	57	80
,, 16	70	76

AVERAGE of each Five Days from June 9th to 24th.

Battery Sample, Value in Dollars.	Salt. Per cent.	Value of Ore in Dollars.	Value of Car Sample Roasted Ore in Dollars.	Soluble Salts. Per cent.	Value of Roasted Ore after Leaching in Dollars.	Value of Tailings from Settlers in Dollars.
112.36	18	136.37	94.27	26	126.88	13.10
130.84	19	160.78	118.34	22	151.74	10.19
125.19	18	153.08	115.01	22	146.99	10.19
		Running half capacity.				
		Average of Six Days, June 9th to 14th.				
114.79	17	137.10	94.27	25	126.88	13.19

The weight of the cast iron of the various parts of the furnace is given below :

	lb.
10 cylinder sections	18,000
4 sole-plates...	2,020
4 chairs	1,880
4 bearing wheels	1,950
1 gear	255
1 pinion	115
4 shafts	1,218
2 pulleys	490
1 feed-pipe	250
8 end-plates	890
Washers and bolts for foundations, bolts and boxes, small bolts	808
Total	27,876
Stetefeldt feeder	3,000

THE THOMPSON-WHITE FURNACE.

This furnace is cylindrical, and of the same diameter throughout its entire length, and is lined with tiles which are made on purpose for it, so that each tile has roughly the shape of a tooth, a hollow corresponding to the projection being at the joint of each brick. The furnace is inclined, and has also the auxiliary fireplace, the ore being discharged automatically at the flue. The angle of the furnace is also adjustable at either end, so that for ores of different characters it may be changed. In order to save the heat lost by radiation, a layer of non-conducting material is placed between the lining on the tiles. The output of the furnace is

regulated by the change of angle more than by the velocity of rotation. The furnaces are of four sizes, varying by 3 ft. from 21 ft. to 30 ft. Their respective diameters are 32 in., 40 in., 52 in., and 60 in.

THE STETEFELDT FURNACE.

The Stetefeldt furnace is one of the most used in the Western States for roasting silver ores. It is composed of a shaft B, Fig. 101, heated by a fireplace G. The flue H is also heated by a fireplace E, and carries off the waste gases and the dust. The object of the furnace is to roast and at the same time chlorurise silver ores. To do this the ore is generally crushed so as to pass a 40 or 30-mesh sieve. This ore, mixed with the proper quantity of salt, is prepared on the charging floor, and is then mechanically discharged into the charging apparatus Fig. 102, which consists of an iron hopper A, placed on the top of the furnace, provided with a draw valve B,which is always open when the furnace is in operation. Above this is another cone, on the top of which is a cast-iron grate C; on the top of this is a screen made of steel plate punched with holes 3 mm. in diameter or larger, as close together as possible. Above the screen is a wrought-iron frame E, on the bottom of which a coarse screen F of heavy wire is placed. This frame with its screen rests on friction rollers G, rotating on brackets H, which can be raised or lowered by means of set screws so as to have any desired distance between the punched screen D and the wire one F. The bracket K carries an eccentric shaft which is connected with the shaft M, which moves the frame E, but as the oscillating motion would not always be sufficient to force the ore through the two screens, stationary blades O are fastened to the brackets N, which can be raised or lowered by the nuts P, so as to bring them into more or less close contact with the screen F. The blades distribute the pulp over the screens evenly. The frame E is kept in motion by a cone pulley, and is so arranged that it can be made to take any motion that it is desirable to give it. The usual velocity is between twenty and sixty strokes per minute. By changing the distance between the screens and also the velocity of the movement, any desired amount of ore can be delivered in the furnace with the greatest regularity. The ore falls into the shaft

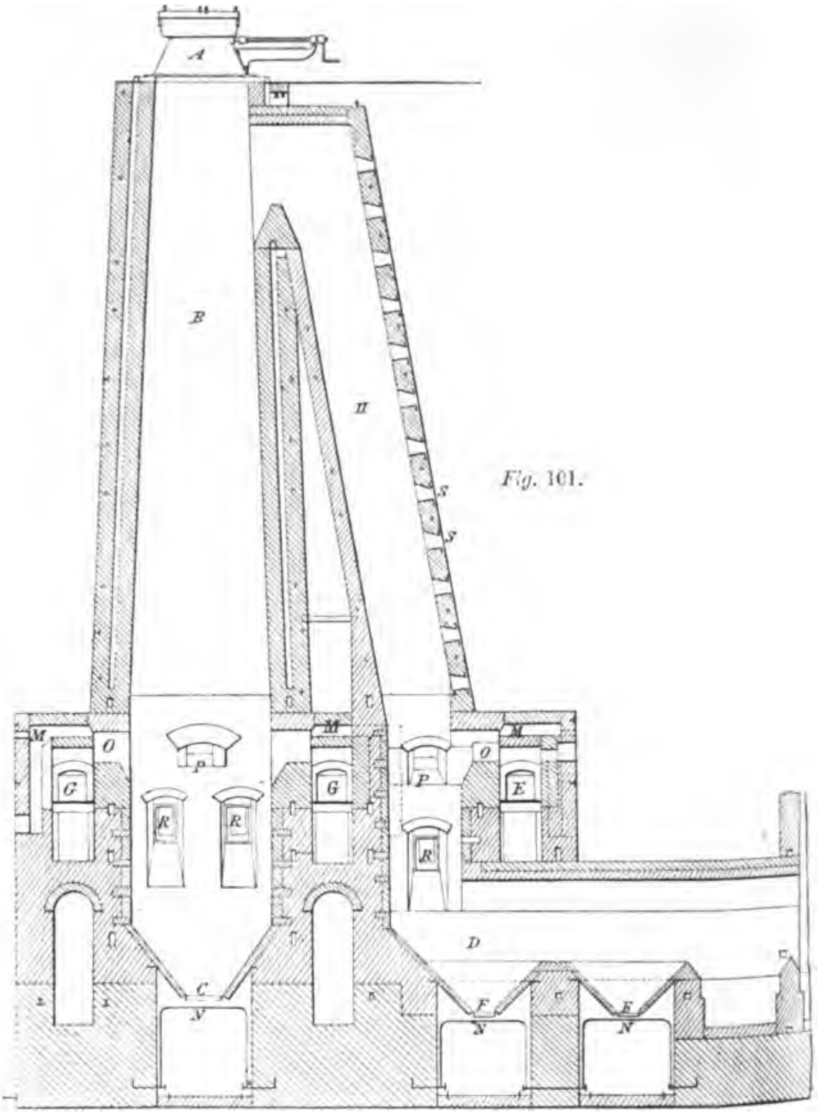
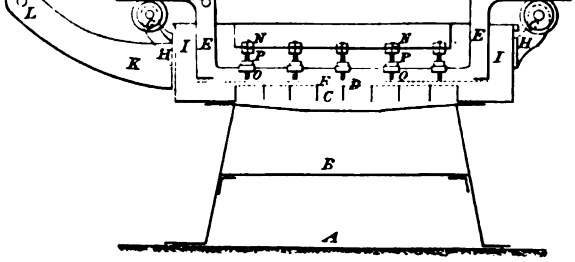


Fig. 101.

Fig. 102.



THE STETEFELDT FURNACE.

B, Fig. 101, where it meets the ascending gases from the two fireplaces G on each side of it. The gases enter the shaft by means of an opening O. Into this same opening a channel M enters, which is connected with the outside air by means of two openings one above and one below, which can be opened as much or as little as desired, to admit the necessary quantity of air for perfect combustion of the gases. The air for the fireplaces is supplied by a door in the ash-pit, in which there is also a damper to allow of admitting more or less of it. At the level of the opening O on each of the other sides of the shaft, there is a door P, which is made both in the shaft B and the flue H to allow of watching the progress of the operation, and below these, both in shaft and flue, are two other doors R, whose lower side slopes towards the bottom of the furnace and allows of admitting tools to scrape the walls when the ore becomes attached to them.

The shaft is usually from 30 ft. to 35 ft. in height, but when the ores are very base, or a large output must be had from a single furnace, it is made as high as 48 ft. It is usually 4 ft. to 5 ft. square. In some of the largest furnaces it has been made as large as 6 ft. square. It is built of two 8-in. brick walls, with a space of 3 in. between them. This space, in order to prevent a loss of heat, is filled in with ashes or sand. This construction is an advantageous one, for when from any cause the fire gets low for a short time, a certain amount of heat is given out from the walls and keeps up for a short period the uniformity of the temperature, leaving a certain interval to apply the remedy. The ore passing through the shaft falls into the hopper C below, which is closed by a valve N, and from this is discharged into the ore cars, and carried to the cooling floor, where it is left red-hot for twelve hours. This always increases the chloruration, and in some ores this increase may amount to as much as 45 per cent. In general the increase is from 5 to 20 per cent. It is desirable to have the temperature of the furnace as high as possible without sintering the ores. It is necessary that this temperature should be kept constant. In order to know exactly what is going on in the furnace, assays must be made for each working shift.

The gases and the dust which is carried with them are discharged into the flue H, in which there are a series of openings S

to clean the flue when necessary, which, with some ores, is as often as once each week. At E there is a supplementary fireplace with openings O and M into the flue. The flame of this fireplace descends through the chamber P R, and is used for completing the chloruration of the dust, which passes down into the chamber D, where it is collected in hoppers F, and is discharged in the same manner as it is from the shaft. The small furnaces when first constructed were discharged by hand, but with the large-sized furnaces now built that is inconvenient, and the hoppers are now always put in.

Formerly the flue at the top of the shaft became so incrusted with the charge as to imedpe the draught and materially reduce the efficiency of the furnace; this has been entirely overcome by introducing doors so that the furnace may be cleaned while in operation. The amount of dust passing down the flue H to be roasted by the auxiliary fire E will vary with the quantity of ore treated in twenty-four hours. When the draught is strong, and the capacity of the furnace large, fully half the ore passes over into the flue. Of this amount about 80 per cent. lodges in the first hopper F of the dust chambers; the rest is carried beyond. The greater the amount of ore treated the less the dust. It will usually vary between 30 and 50 per cent. The dust chambers are not discharged frequently, and the last ones are often left for a week. Being thus piled up and at a high temperature, the same effect is produced as on the cooling floor, so that it has the full benefit of the extra chloruration. From the dust chambers the gases pass through a chimney which should be from 50 ft. to 100 ft. above the top of the shaft. For small furnaces the horizontal section of the chimney is generally 4 ft. square; for very large ones 5 ft. The best disposition is to build the furnace at the bottom of a valley or cañon with a flue leading up to a chimney on the top of a hill.

The draught is regulated by a damper at the end of the dust chambers, which diminishes the size of the flue leading to the chimney. The amount of ore which will fall through a given distance may be very materially diminished by increasing the draught as it rises against the falling particles. More dust in this case will pass over into the flues. It is necessary to ascertain

what quantity of air is best suited to the treatment of any ore and then maintain it. If there is not enough, which may be caused by obstructions in the flues or cracks in the furnace, the ore will not be sufficiently acted on, especially in the shaft. The chloruration will consequently be low. If there is too much there is danger of the flues becoming choked, as well as of the ore being imperfectly roasted. As there are doors for observation in all parts of the furnace, any change may be seen at once, its cause ascertained, and the remedy applied.

The ore and salt are sampled every half-hour by the fireman at the feeders where they are thoroughly mixed. The samples are taken from the roasted ore, when it is discharged into the cars, with a tube thrust to the bottom of the car; each car from the flue and the furnace is thus sampled as it goes to the cooling floor. All the samples are then put together and tested. After the ore has remained for twelve hours or more on the cooling floor it is again sampled in the same way with the tube. This gives the difference in chloruration between the two. When it is necessary to ascertain how the furnace or the flue is working, a sample is taken by introducing a tool or a piece of sheet iron and allowing the falling ore to accumulate upon it. As a general thing it will be found that the chloruration is less in the samples taken directly from the furnace, than in those taken from the cooling floor.

It has always been supposed in the amalgamation of roasted ores, that all the silver which was present as chloride was attacked by the mercury and extracted. The fact that the extraction of the silver is greater than the amount of silver shown by the assay to be chlorurised, has led to this conclusion. Mr. Russell, at the Ontario Mill, has made the following investigations to show that this is not the case. The average chlorurations and amalgamation for a period of nine months, from April to December, 1881, were as follows:*

	Furnace No. 1.	Furnace No. 2.
Chloruration	89.5	90.2
Amalgamation	91.6	90.1

The tests were the results of 448 car samples, and all the results

* "Engineering and Mining Journal," vol. xxxiv., p. 256.

were in duplicate. The examination shows that from 20 to 50 per cent. of the silver contained in them was chlorides.*

	Furnace No. 1. Silver.	Furnace No. 2. Silver.
Remaining in tailings ...	8.4 per cent.	9.9 per cent.
Present as chloride of silver	3.0 ,,	3.6 ,,

If this silver had all been amalgamated, the amalgamation would have shown 94.6 from the first furnace, and 93.7 from the others. After roasting in the shaft furnace up to the point when the heat would about sinter the ore, the chloruration of the tailings varied between 4.3 and 22.3 as compared with 19.4 and 55.1. This was probably owing to the decomposition in the hot charge, and the cooling down with water spray which disintegrated the lumps.

The capacity of a Stetefeldt furnace varies very greatly with different kinds of ores, according as they contain more or less base metals, and are consequently more or less difficult to roast. It may vary from single or double to triple, according as the ore contains less sulphur. It also varies slightly with the position of the furnace with regard to its chimney, so that the draught may be greater or less. What can be absolutely obtained from a given furnace with the most favourable conditions of ore and draught has not yet been determined. At the Ontario Mill, Utah, as much as 65 tons have been roasted experimentally in one furnace in twenty-four hours, and at the Northern Belle, Nevada, their usual output is about 55 tons every twenty-four hours, but as much as 70 tons have been put through in the last-named mill. When the maximum is reached, it will also be with ores containing little sulphur. In the last case the ores were almost entirely oxidized, the gangue being oxides of iron and manganese, while at the Lexington Mill, Montana, the ore is compact pyrites with zinc blende and galena, and only 30 tons can be roasted in the same-sized furnaces in twenty-four hours. Most of the furnaces, however, are limited in their capacity by the output of their stamp mills, which is very often much below the capacity of the furnace. When the roasting is properly done, the very short time which the ore comes in contact with the fire

* "Engineering and Mining Journal," vol. xxxiv., p. 256.

is a great advantage, so far as the loss in silver is concerned. The amount of silver lost in roasting is very much larger than is generally supposed,* and will be all the larger if it is associated with volatile compounds like arsenic, antimony, zinc, &c. With a good set of dust chambers, the loss actually found in the Stetefeldt furnace is very much smaller than that which is usually found in roasting in reverberatory furnaces.

The size to which the ore should be reduced depends principally on the condition of the minerals in the gangue. If they are disseminated through a hard rock, it may be necessary to make the pulp pass a 60-mesh screen; if disseminated in larger grains they must be made coarser. What must be done in each case can only be decided by experience. It must be borne in mind that crushing so that the ore will pass through a screen with meshes of given size does not imply that the ore is not crushed finer. It has been shown† that of ore which passed a 20-mesh screen, only 13.8 per cent. remained behind on a 40-mesh screen. Of ore that was crushed through a 30-mesh screen only 8 to 10 per cent. remained on a 40-mesh screen, and so on. For this reason batteries having fine screens will always crush less than those having coarse ones. There is, however, a limit to the extent of coarse crushing for perfect roasting. In amalgamation this depends on the impossibility of washing heavy pulp out of the settler without losing mercury. The extreme limit appears to be about a 30-mesh screen. The 26-mesh screen used at the Ontario Mill had to be abandoned on account of the difficulty in working the settler. The percentage of amalgamation was, however, about the same as shown by the figures below, representing a six weeks' run, two weeks' trial being given to each kind of screen.

	Furnace No. 1, Amalgamation.	Furnace No. 2, Amalgamation.
No. 30 screen	89.9	89.7
,, 26 ,, ...	90.0	91.1
,, 30 ,, ...	90.4	90.2

The size to which the ore is to be crushed before roasting will also depend on whether it is to be subsequently treated by lixiviation or amalgamation. Experience has shown that while

* "Treatment of Flue Dust at Ems," Trans. Am. Inst. Min. Eng., vol. xii.

† "Engineering and Mining Journal," vol. xxxv., p. 348.

there are fixed limits for amalgamation, those required for lixiviation are much coarser, and that the exact limit must be determined in each special case, but it is evident that the particles of precious metal can be much more easily reached in coarsely crushed ore by lixiviation than by amalgamation. It has been found by experience that ordinary ore which passes a 30-mesh screen would be as well roasted as that from a 40, but it might in certain cases with ores containing a great deal of sulphur, which are called "heavy ores," not be advisable to crush it coarse, so that it is usually crushed fine. The following results of the effect of crushing through screens with meshes of different sizes were obtained in 1879.*

	50-mesh Screen.	30-mesh Screen.
Value of roasted ore	143.29 dols.	117.75 dols.
,, tailings	13.42 ,,	12.62 ,,
Chlorurations	90.6 per cent.	89.1 per cent.
Silver extracted by amalgamation	90.6 ,,	89.3 ,,

The average amount of ore crushed in a 20-stamp mill and roasted in twenty-four hours was 35.5 tons.

The quantity of salt required to perform the chloruration depends on the character of the ore treated, and will generally vary from 2.5 to 18 per cent. That there is an advantage in increasing the amount up to a certain point is shown by the following results of experiments made at the Ontario Mill.†

Roasted with	Silver Chlorurised
2 per cent. salt	44.5 per cent.
4 ,,	52.0 ,,
6 ,,	60.4 ,,
8 ,,	76.0 ,,
10 ,,	82.8 ,,
12 ,,	88.4 ,,
14 ,,	90.9 ,,
16 ,,	93.0 ,,

Exactly what the best quantity is must be determined by trial in each special case. It is not advantageous, as in the reverberatory furnace, to use an excess of salt, as it is not only thoroughly mixed with the ore beforehand, but each particle of ore during the whole of its fall in the furnace, is exposed to the chlorurising influences, and again both when it lies in the hopper at the

* Communicated by Mr. Stetefeldt.

† Russell's Improved Method, Trans. Am. Inst. Min. Eng., vol. xiii.

bottom and on the cooling floor. The quantity of salt which will give the highest chloruration having been determined, it is wasteful to add more, as it has been shown by experiments made at the Ontario Mill that 3 per cent. of undecomposed salt is contained in the roasted ore, more being present in the samples from the shaft than from the flue. The volatilisation of the salt is therefore not as great in this furnace as is usually supposed. The same investigation showed that copper and zinc were found in the shaft only as chlorides, while in the flue they were present exclusively as sulphates, and that this is probably the reason why the chloruration is usually higher in the flue than in the shaft. While some small quantity of sulphides was found in the roasted ores of the shaft, none at all were found in the ore from the flue.

What the chloruration will be in the case of each ore depends, as will be understood from what has gone before, on a number of circumstances. These are in general the kind and quality of the ore, especially the quantity of sulphur it contains, the velocity of feeding, the strength of the draught, the quantity of salt, the heat of the fire, and the time the ore remains in a heap after being discharged. Any or all of these may prevent the proper action of the salt on the ore. In the early days of the furnace, the practice was to draw the pulp at once and cool it with water, and send it immediately to the pans. In these cases the ores were of very favourable composition, and did not require to be left on the cooling floor, although this treatment would have raised the chloruration several per cent. The following Table shows the early results obtained at the Manhattan Mill:

		Per Cent. of Chloruration.
September, 1874...	...	88.8 (shaft and flue together)
October ,, ...	...	90.3
November ,, ...	...	89.2
December ,, ...	...	91.1
January, 1875...	...	89.8
February ,, ...	...	90.0
March ,, ...	...	90.6
April ,, ...	...	90.8
May ,, ...	...	91.6
June ,, ...	...	92.8
July ,, ...	...	92.3
August ,, ...	...	91.8
September ,, ...	...	91.5

Below is given the average workings of the Manhattan Mill for each of the four weeks of November, 1877:

—	Day. Per Cent. of Chloruration.		Night. Per Cent. of Chloruration.		Hopper. Value of Ore.		Tailings. Value.		Per Cent. left in Tailings.	
	Stack.	Flue.	Stack.	Flue.	Day.	Night.	Day.	Night.	Day.	Night.
					dols.	dols.	dols.	dols.		
1st week	89.5	91.3	90.8	91.1	180.86	183.10	10.77	11.44	6.4	6.4
2nd ,,	90.9	91.3	91.1	91.0	199.71	208.24	14.35	10.05	7.1	4.9
3rd ,,	89.9	89.9	89.5	90.3	177.52	153.47	14.29	13.47	7.9	8.7
4th ,,	91.1	92.7	90.7	91.5	221.70	220.08	12.00	13.24	5.4	6.5

The following Table* gives the result of the working of the Surprise Valley Mill in 1875:

SURPRISE VALLEY MILL AND WATER COMPANY'S MILL, PANAMINT, CAL.

—	Roasted Ore from Shaft. Day.		Roasted Ore from Flue. Day.		Roasted Ore from Shaft. Night.		Roasted Ore from Flue. Night.		Battery Sample.	Tailings.
	Assay Value per Ton.	Chloruration per Cwt.	Assay Value per Ton.	Chloruration per Cwt.	Assay Value per Ton.	Chloruration per Cwt.	Assay Value per Ton.	Chloruration per Cwt.	Value per Ton.	Value per Ton.
										dols.
Week ending September 28	80.26	93.36	89.21	94.89	83.40	93.32	89.68	94.65	93.70	7.36
Week ending October 16...	77.18	92.4	87.56	92.90	77.42	93.0	84.82	93.9	88.25	8.63
Week ending October 23...	68.66	91.7	78.98	92.9	70.68	92.4	78.54	93.2	79.12	7.60

The following Table† gives the values and per cent. at the Lexington Mill:

* Communicated by Mr. Stetefeldt. † *Ibid.*

—	Furnace.	Chlorura-tion.	Left in Tailings.
	No.	Per cent.	Per cent.
December, 1882 . .	1	82.9	10.0
,, ,, . .	2	85.1	11.8
January, 1883 . .	1	85.9	9.9
,, ,, . .	2	85.9	10.8
February ,, . .	1	86.9	7.4
,, ,, . .	2	86.0	8.5
March ,, . .	1	89.9	7.8
,, ,, . .	2	90.1	8.9
April ,, . .	1	92.6	6.1
,, ,, . .	2	92.5	6.5
May ,, . .	1	93.8	5.5
,, ,, . .	2	94.6	5.3

The following Table gives the workings of the Lexington Mill for four months.

—	Battery Sample.		Roasted Ore.		Tail Samples.		Per Cent. in Tailings.		Per Cent. of Salt.	Per Cent. of Soluble Salt.	Per Cent. of Chloruration.
	Silver.	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.	Gold.			
	oz.	oz.	oz.	oz.	oz.	oz.	oz.	oz.			
April . { E. *	45.2	.76	47.6	.76	2.9	.347	6.1	45.7	14.8	21.1	92.6
W.†	44.5	.75	47.5	.76	3.0	.35	6.5	46.5	14.6	20.3	92.5
May . { E.	57.0	.85	52.7	.88	3.0	.386	5.5	43.8	14.5	19.5	93.8
W.	50.5	.81	52.9	.83	2.9	.385	5.3	46.4	14.1	19.9	94.6
June . { E.	51.3	.81	54.5	.83	4.2	.38	7.8	46.0	14.5	19.0	89.2
W.	51.1	.80	55.2	.84	3.7	.38	6.8	45.3	14.5	20.0	92.3
July . { E.	51.9	.79	51.3	.81	5.7	.37	9.7	45.2	12.6	16.7	92.0
W.	50.8	.80	54.0	.81	4.7	.37	9.2	45.4	13.2	17.2	92.0

The following Table‡ gives the average chloruration tests per month at the Ontario and Manhattan Mills at different periods, all the ore being left on the cooling floor for twelve hours before the assay was made:

* E. = East furnace. † W. = West furnace.

‡ "Engineering and Mining Journal," vol. xxxv., p. 377.

	Ontario Mill, March to Sept., 1882.	Manhattan Mill, Oct. 1, 1874, to Sept. 30, 1875.
	Per cent.	Per cent.
October	...	90.3
November	...	89.2
December	...	91.1
January	...	89.8
February	...	90.0
March	92.0	90.6
April	91.3	90.8
May	92.9	91.6
June	92.5	92.8
July	93.0	92.3
August	92.2	91.8
September	91.4	91.5

The following Table gives the workings of a new Stetefeldt furnace; average of each five days from May 22 to June 30:

Battery Sample, Value in Dollars.	Salt per Cent.	Value of Ore in Dollars.	Value of Car Sample Roasted Ore in Dollars.	Soluble Salts per Cent.	Value of Roasted Ore after Leaching with water, in Dollars.	Value of Tailings from Settlers, in Dollars.
111.61	19	138,14	104.07	29	146.57	13.19
110.11	19	135,57	106.33	26	143.36	14.70
105.58	20	132,03	119.92	27	137.92	10.93
110.48	18	134,46	107.09	26	145.49	11.68
115.03	18	140,45	111.23	26	150.30	12.06
121.79	18	148,87	114.63	27	156.12	12.81
134.49	18	165,36	124.06	27	169.64	13.94
115.76	18	141,87	108.97	27	148.90	11.68

The following Table* shows the comparative results of the chloruration in three different mills working on different ores:

	Ontario Mill.		Manhattan Mill.	Surprise Valley Mill.
——	58 Samples, July, 1881.	131 Samples, October 12, 1882, to January 1, 1883, Russell's method.	198 Samples, November 1 to December 24, 1877.	96 Samples, September 19 to October 23, 1875.
Per cent.	Times.	Times.	Times.	Times.
85	...	...	...	1
87	...	1	...	1
88	1	...	7	...
89	1	...	28	1
90	2	2	49	3
91	8	10	68	7
92	13	30	28	18
93	21	46	15	24
94	9	33	...	26
95	3	9	3	12
96	...	...	...	3

* "Engineering and Mining Journal," vol. xxxv., p. 377.

The following Table gives the results obtained at the Ontario Mill with Russell's process.

Number of Screens used in Crushing	Per Cent. of Salt used in Roasting.	Value of Roasted Ore in Ounces.		Per Cent. of Silver extracted by Chloruration Test with Sodium Hyposulphite.		Per Cent. of Silver extracted by Lixiviation with Russell's Extra Solution.		Per Cent. of Silver extracted from a Charge of 2 Tons by Russell's Lixiviation Process.
		Ore from		Ore from		Ore from		
		Shaft.	Dust Chambers.	Shaft.	Dust Chambers.	Shaft.	Dust Chambers.	
30	9.0	58.4	73.0	92.0	92.0	95.2	94.1	96.2
20	12.5	89.2	108.0	92.0	93.1	97.0	97.1	97.0
16	12.0	76.8	91.1	91.0	93.0	96.0	96.5	97.0
16	18.0	63.2	78.4	88.0	93.0	95.2	97.0	97.5
16	16.0	58.3	72.3	89.0	94.7	93.5	95.6	95.0
16	7.5	82.2	105.5	79.7	89.0	88.4	91.0	91.1
16	8.0	60.0	93.0	82.0	88.0	88.5	90.3	91.0

From all these experiments it appears that the chloruration is less in the shaft than in the flue; that there is a great gain in leaving the ore a certain number of hours in a hot heap before cooling; that the extraction by lixiviation is generally greater than by amalgamation; that if the process is by lixiviation, the best results are obtained by the use of Russell's process.

The amount of labour about this furnace is extremely small. A single man per shift is all that is required to look after the fire and the feeding machine at the top, and take the regular samples from the pulp. For this purpose a lift is provided to take him from the bottom to the top of the furnace. For discharging and cooling the roasted ore two men per shift of twelve hours are required. This is supposing that the furnace is running from 20 to 25 tons per twenty-four hours. For furnaces of larger capacity the firemen as well as the pulpmen should run eight-hour shifts, so that the labour of nine men in all is required in twenty-four hours.

The usual fuel is wood. The quantity of it used in twenty-four hours will depend upon the character of the ore, the size of the furnace, and the quantity of ore that it is desired to treat in that time. It may be said, in general terms, that the larger the furnace the less fuel per ton of ore will be used. Highly sulphuretted ore requires less fuel because the sulphur in the ore

burns, but the output is diminished. Oxidized ores, on the contrary, require more fuel, but the output of the furnace is very largely increased. The consumption of fuel will generally be from 1½ to 5 cords per furnace in twenty-four hours for an output of under 40 tons, the maximum being for oxidized ores, the minimum for sulphuretted. It is desirable that the wood used for the purpose should be well seasoned, in order to use less of it.* The amount which is generally used is from 2½ to 2¾ cords of good and well-dried wood in twenty-four hours, or its equivalent in other fuel, to treat 20 to 25 tons of ore in that time. When more than this is treated more fuel will be required. In the largest-sized furnaces four cords are used. At the Manhattan Mill in Austin, Nevada, and elsewhere, gas producers are used with excellent results, in which a mixture of charcoal and wood is burnt, but their construction and management are too costly and complicated for general use.

The details of the materials required in the construction of the furnace given below, refer to one capable of treating from 40 to 100 tons in twenty-four hours.

Stone for foundations of furnace and dust chamber	3000 to 4000 cubic feet
Common brick for furnace, flues, chimney, and dust chamber ...	250 to 275 M.
Firebrick	5 M.
IRONWORK:	lb.
Feeder	2,800
Plain castings	12,000
Finished "	4,000
Forged and boilermaker work	3,850
Plain wrought iron	850
Brick stays	9,500
Bolts	4,500
Rails for braces	10,000
	47,500

This includes all plates, screens, tools, and discharge cars. The latter cost from $45 to $50 each.

The expenses of treating 16 tons in twenty-four hours of ore of all kinds and grades at Reno† were:

* Trans. Am. Inst. Min. Eng., 1884. Proc. Am. Soc. Civil Eng., 1881.
† "Mineral Resources of the United States, 1876," p. 412.

	$
2 firemen, at $2.00	4.00
4 feeders and dischargers at $1.75	7.00
2560 lb. of salt (8 per cent.) at 1½ cents	38.40
1½ cords of wood (slabs) at $5.00	7.50
	56.90

Or per ton of ore, $3.53

The whole cost of the treatment of the ore, including the amalgamation at Reno, was per ton:

	$
General expenses	2.40
Hauling	.92
Sampling and crushing	1.52
Assaying	1.23
Roasting	3.53
Amalgamating...	4.37
	13.97

Recently the total expenses have been very much diminished in Nevada, as shown below.

	$
2 firemen, at $4.50	9.00
4 pulp coolers, at $4.00	16.00
2¾ cords of wood, at $8.00	22.00
Wear of screens	1.00
Total labour and fuel for 25 tons in 24 hours	48.00
Labour and fuel per ton	1.92
7 per cent. of salt at $40 per ton	2.80
Total expense of chlorurising and roasting	4.72

The furnace at first sight appears complicated, but the management is extremely simple, and is easily learned by an ordinary labourer, much less judgment being required with it than with a reverberatory furnace, where the men must judge from the sight of the ore as to the condition in which it is. Here, the operation being entirely mechanical, the ore charger being once set at a given velocity, very little judgment is required of the men. When the draught is good there is no danger, as in the reverberatory furnace, of sintering the ore even if it contains considerable lead.

The furnace is adapted to ores of almost any grade. Those at Reno treated ores of from 30 oz. to 800 oz. The Manhattan treats only ores above 100 oz.; generally it treats only high-

grade ores. The Lexington furnace treats ore containing from 40 oz. to 60 oz.; the Ontario from 60 oz. to 100 oz.; and the Northern Belle from 30 oz. to 100 oz.

Though invented so many years ago, the furnace has up to the present time only been used for the chlorurising roasting of silver ores. It appears, however, as though it might be used for any other purposes requiring an oxidizing roasting except in such cases where it is necessary to manufacture sulphuric acid from the escaping gases, in which case it cannot be used. Wherever it is applicable, it has the advantage of durability and simplicity of construction, as it occupies less space than the ordinary roasting furnaces. It can be so worked that almost the whole of the pulp entering the furnace can be regained, even the dust being well chlorurised, so that the loss in silver is very small. It uses but little power and a very small amount of labour per ton of ore treated. The amount of fuel required to roast a ton of ore is also very small, while the output of the furnace is very large, and can be easily increased by working it with more shifts and making them shorter.

CHAPTER VI.

THE PATIO AND CAZO PROCESS.

THE various amalgamation processes depend on bringing the ore into contact with mercury. It is necessary that the ores so treated by amalgamation should be very finely crushed. To be carried out cheaply the work should be done on a very large scale, and hence every amalgamating works must be accompanied by some kind of a crushing mill, of sufficient capacity to crush a little more than the daily needs of the amalgamation plant.

The amalgamation processes in use in the United States for the extraction of silver are: 1. The Patio. 2. Barrel amalgamation. 3. Pan amalgamation. The conditions of climate, where part of the silver ores are found, has forced the almost entire abandonment of the *patio* process, but it may possibly come into use again, in the hot portions of the country. Barrel amalgamation has also been given up for the reasons stated in that article. Taking into consideration the fact that most of the American mills have been started by mechanics, and superintended by men who are generally innocent of all metallurgical knowledge, it seems wonderful that they should be able to work the ore as closely as they do. It seems to Europeans very extravagant that tailings in certain districts contain from 15 oz. to 18 oz. of silver to the ton, or even more; but this is generally the case, because the price of labour is so high that it will often cost $21 to get out the $15. To attempt to use any process based on European experience of labour would, therefore, be unwise in the West. The doing of it, however, is the reason why so many foreign enterprises have resulted disastrously.

The process of amalgamation which is still used both in Mexico and Chili, is called the Patio or American method of amalgamation, in order to distinguish it from the process used so long at Freiberg, known as the "Freiberg barrel amalgamation," and that which has now for so many years been almost exclusively used in the western part of this country, known as "pan amalgamation." It is effected in two different ways, according to the country in

which it is used. In Mexico it is called the Mexican or Patio method, and in Chili it is known as the Chilian or Cazo method. These processes do not differ essentially, except in the mechanical appliances which are used for carrying them on. The Patio method was, until questioned by Dr. Percy,* supposed to have been invented in Mexico, about 1557, for beneficiating the silver ores which occur there.

The Cazo method was invented in 1609, in Peru,† and has not been used much except in South America and Mexico. The Patio method is used in Mexico on ores that have a mean yield of from $30 to $60 to the ton. Ores of much higher grade than this are treated, provided they are not refractory, but when they are rebellious they are generally treated by fusion. In order to do this, however, the yield must be large, for fuel is very dear on the plains of Mexico.

It is quite rare that anything is done to the ores before treatment except hand-picking to sort out those of high grade from those of less yield, and to remove some of the sterile material. Occasionally, however, they are treated in a rude way. At Zacatecas‡ very impure ores are broken by hand into small pieces, made into a pile surrounded by a rude wall laid up dry, and imperfectly roasted with charcoal. In the districts of Tasco and Sultepec, where sulphurous ores are abundant, they are roasted with wood in the same furnace, *comalillos*, in which the magistral is made, but not efficiently, though the operation lasts twelve hours. The *colas*, the concentrated sulphides, are also roasted in piles. This pile-roasting is not only very insufficiently done, but is very uncertain in its results. The object is to remove the substances which attack the mercury, but, owing to defects both of fuel and arrangement of the pile, but little else results from it than the blind following of a routine, which has no other reason than that it has been practised somewhere else. There is always danger that, in roasting these ores, the heat will be raised sufficiently high to melt them. When they are rich, a fusion treatment is much more rational. It is doubtful whether, with a dear fuel, much is gained by roasting previous to the treatment on

* Percy's "Silver and Gold," Part I., p. 562, London, 1880.
† *Ibid.*, Part I., p. 656.
‡ Phillips' "Gold and Silver," p. 352, London, 1867.

the *patio*. It is generally the gangue which determines the name of the ore, but it is sometimes called after its size. Quartz is called *guija*, and quartzose ore, *guijoso*; feldspar is called *caliche*; feldspathic ores, *calichoso*. When there is much gangue it is said to be *desploblado*. *Quemazon* is a black porous decomposed ore. Large pieces of the first and second-class ores are called *gabarro*. The smalls are called *metal granza*. The minerals which are usually found as ores, or associated with them, are native silver, *plata*; kerargyrite, *plata cornea blanca*; embolite, *plata cornea verde*; bromyrite, *plata verde*; iodyrite, *plata cornea amarillia*; argentite, *plata negra*; ruby silver, *rosi clara*; arsenopyrite, *ferro blanca*; galena, *plomo*; and zinc blende, *copelilla*.

The ores are generally distinguished as of two kinds, the black, *negros*, and the coloured, *colorados*. The former are found in the lower part of veins, and comprise all the ores containing sulphur. The *colorados* are found in the upper parts of veins, and are composed generally of the iodides, bromides and chlorides, with some native silver mixed with them. The gangue is generally oxide of iron, carbonate of lime, or quartz; occasionally some argillaceous schists which, when they are not attacked by the reagents, can be as easily treated as the others. This method is the only one that can be used in many places in Mexico, on account of the high price of fuel. In some places the very rich rebellious ore is roasted and then treated, and this should always be done with all the *negros* when fuel is not so dear as to render such a treatment impossible. When the gangue is attacked to any extent this process cannot be used. The works where these operations are carried on are called *haciendas*.

The process consists of five different operations.

1. Crushing the ore in a Chilian mill, stamp, or crusher.
2. Grinding and amalgamating the ore in an arrastra.
3. Treatment on the *patio*.
    - *a.* Making the *torta*.
    - *b.* Introducing the reagents.
    - *c.* Separating the amalgam.
4. Treatment of the amalgam.
5. Refining the silver.

The following tree* gives a very accurate idea of all its details.

* Patio Process, San Dimas, Trans. Am. Min. Eng., vol. xi., Pl. I.

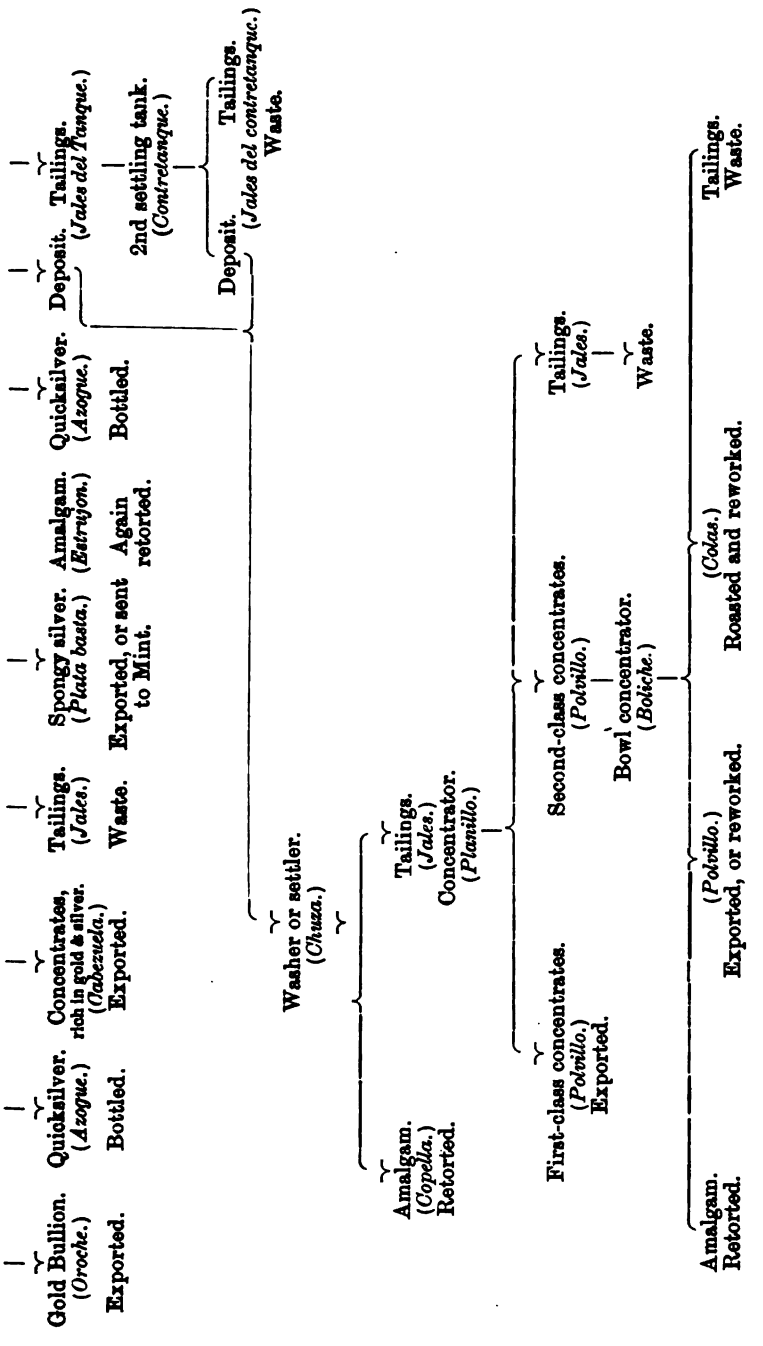
SCHEME OF PATIO PROCESS.

MINE.

- **First-class ore.** (*Metal de Exportacion.*) Exported.
- **Second-class ore.** (*Metal de Beneficio.*)
  - **Crusher.** (*Quebradero.*) → Arrastra.
- **Third-class ore.** (*Granza de Llunque.*) → Arrastra.
- **Fourth-class ore.** (*Tierras de Labor.*) → Arrastra.

Arrastra. (*Tahoma.*)

- **Scrapings.** (*Raspadura.*)
  - **Washer or settler.** (*Chuza.*)
    - **Amalgam.** (*Copella.*)
      - **Retort.** (*Quemadero.*)
    - **Tailings.** (*Jales.*)
      - **Concentrator.** (*Planillo.*)
- **Slimes.** (*Lama.*)
  - **Patio.**
    - **Washer or settler.** (*Lavadero.*)
      - **Amalgam.** (*Copella.*)
        - **Retort.** (*Quemadero.*)
      - **Tailings.** (*Jales.*)
        - **Settling tank.** (*Tanque.*)


Gold Bullion.
(Oroche.)
Exported.
Quicksilver.
(Azogue.)
Bottled.
Concentrates, rich in gold & silver.
(Cabezuela.)
Exported.
Tailings.
(Jales.)
Waste.
Spongy silver.
(Plata basta.)
Exported, or sent to Mint.
Amalgam.
(Estrujon.)
Again retorted.
Quicksilver.
(Azogue.)
Bottled.
Deposit.
Tailings.
(Jales del Tanque.)
2nd settling tank.
(Contretanque.)
Deposit.
Tailings.
(Jales del contretanque.)
Waste.
Washer or settler.
(Chuza.)
Amalgam.
(Copella.)
Retorted.
Tailings.
(Jales.)
Concentrator.
(Planillo.)
First-class concentrates.
(Polvillo.)
Exported.
Second-class concentrates.
(Polvillo.)
Bowl concentrator.
(Boliche.)
Tailings.
(Jales.)
Waste.
Amalgam.
Retorted.
(Polvillo.)
Exported, or reworked.
(Colas.)
Roasted and reworked.
Tailings.
Waste.

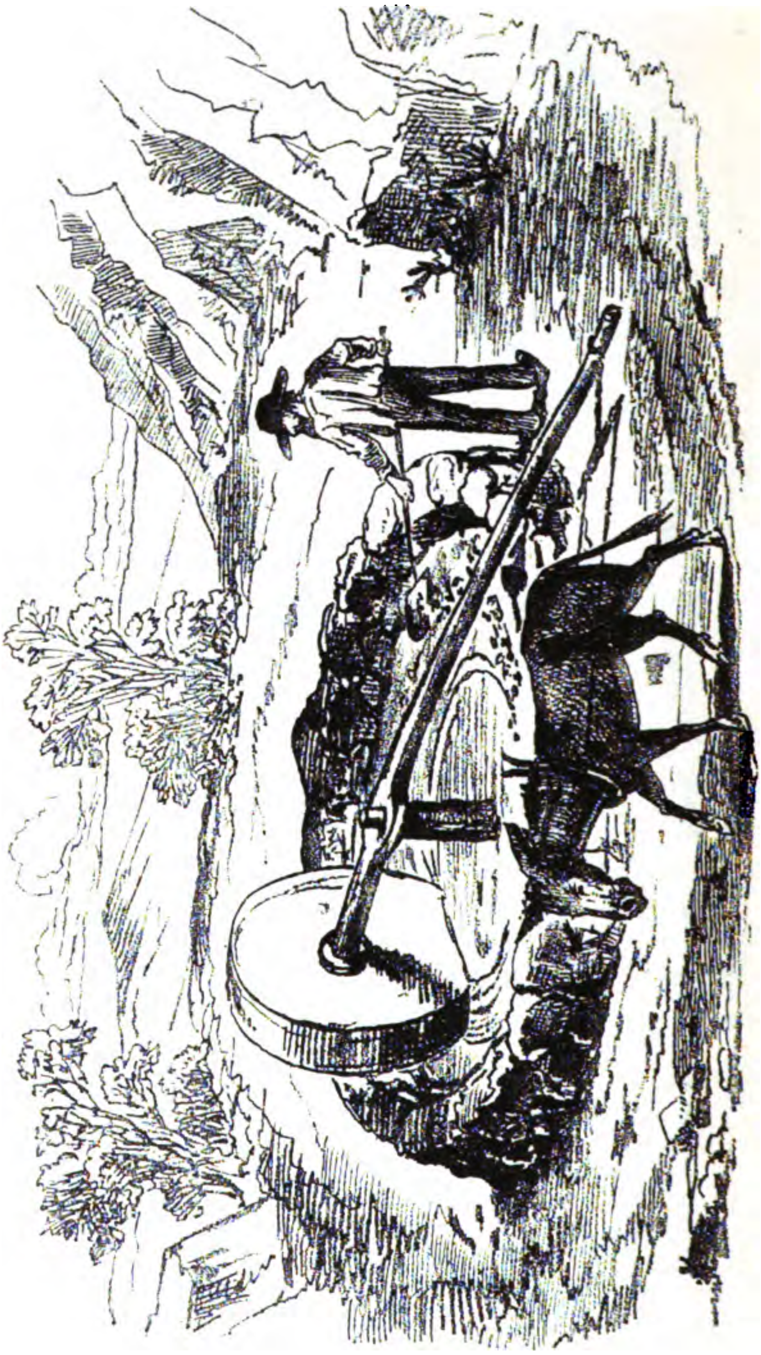


Fig. 103. Old Chilian Mill.

I. Crushing the Ore.

The amalgamating works consist of a large court, *patio*, surrounded by sheds, *galera*, in which the apparatus for comminuting the ores is placed. All of the ore to be treated must be reduced to an impalpable powder. The court is always paved in some way, generally with stones; and if it is desirable that it should contain only a single pile, it is only 10 or 15 metres square. When a number of piles are made in the same inclosure, it must be very large, as the piles are often 7 or 8 metres in diameter. The court then would be 50 or 60 metres square, or even larger.

The ores are generally sorted, according to their silver contents and gangue, into three or four classes. At San Dimas* there are four grades. The first is the lumps of pure ore picked out by hand, *metal hecho*, or made ore, free from gangue, worth $400 or more to the ton. This is called *metal de primera classe*, or *metal de exportacion*. The second is ore for the patio, called *metal de beneficio*. It differs from the first only in being of less value, and by having gangue mixed with it. The third class embraces the smalls from the hand-picking, and varies in value according to the value of the ores from which it is selected. It is called *granza de llunque* or *tierras de llunque*. The fourth class comprises the smalls from the mine. It is mixed with much gangue and dirt. It is called *granza de labores* or *tierras de labores*. At Chihuahua,† where the ores are almost entirely composed of native silver in a calcite gangue, they are separated into three classes; the first containing more than $2500 to the ton; the second, more than $1000 and less than $2500; the third class, under $1000 and averaging about $250.

This classification, however, differs at every works; the first consideration being always the value of the ore; the second, the kind and quantity of gangue, according as it may or may not be attacked by the reagents; and lastly, the size of the pieces.

At Chihuahua the third-class ore is carried to the stamps; *morteros*, the best ore, is carried to the store-house, from which it is weighed out. The ore is crushed with small stamps, weighing about 150 kilos each, with a fall of 0.20 m. to 0.25 m. The

* Patio Process at San Dimas. Trans. Am. Min. Eng., vol. xi.

† Report United States Mining Commissioner, 1874, p. 435.

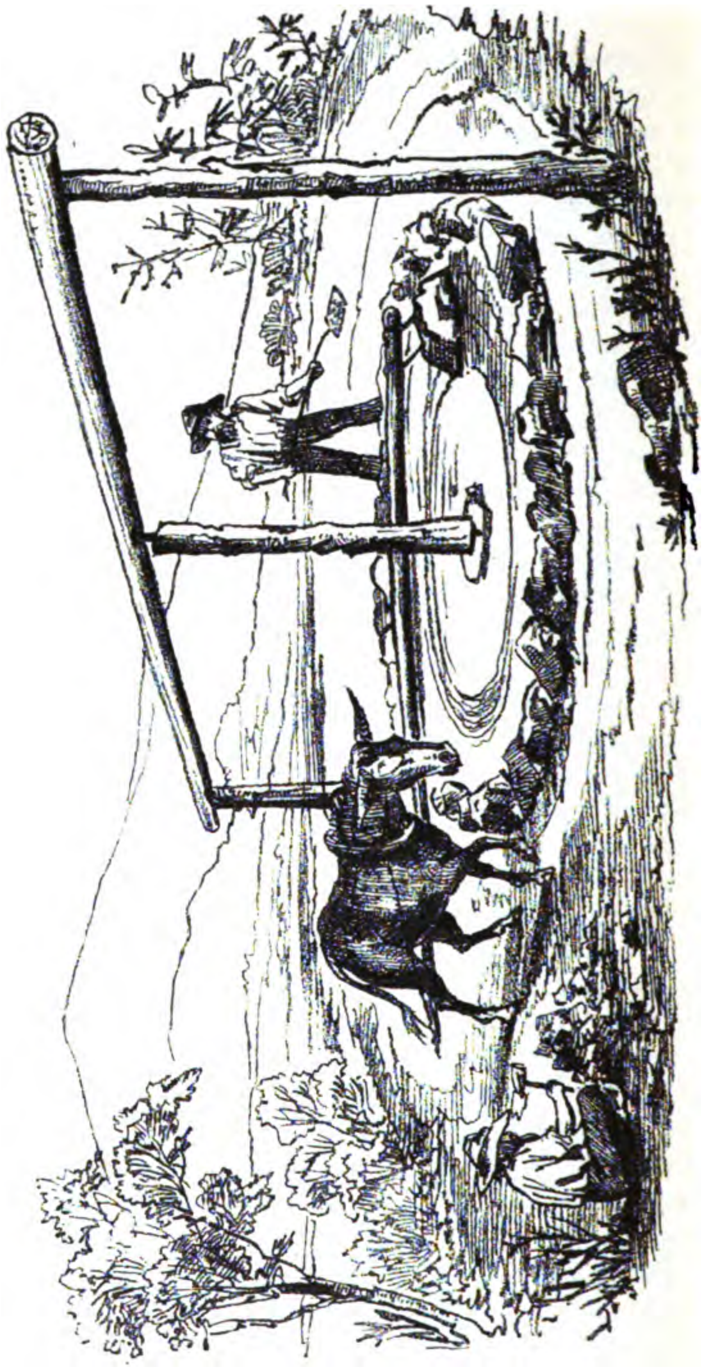


FIG. 104. MEXICAN ARRASTRA FOR GRINDING ORES.

slots of the screens are 0.15 m. wide. The coarse sand which passes the screens is called *granza.* The lumps of native silver which do not pass through the screens are cleaned by hand. The stamps are run by a horizontal water-wheel. From the stamps the ore is taken to the arrastra. At San Dimas, and generally in the whole country, the ore which is not small enough is broken by hand until it is small enough to be stamped in old-fashioned German stamps, with wooden stems and iron shoes. Three stamps, weighing 0.50 kilo. each, with a drop of 0.22 m., are capable of crushing and forcing out of a screen with 0.15 slots, about 8 tons in twenty-four hours. These stamp mills, *molinos,* are sometimes run by mule power. In some works rolls are used.

The Chilian mill, *trapiche,* the primitive form of which is shown in Fig. 103, is also still in use. This was made of a large stone, generally granite, about two metres in diameter, with an edge of 0.40 m. wide, weighing between three and four tons. It was made to revolve on the circumference of an inclosure roughly built up of stones, on a long horizontal arm, which pivotted on a heavy piece of metal set in a post, driven into the ground in the centre of the grinding space. When metal was not easily procured, the beam was made to turn on a piece of tough wood. The stone revolved on one end of this beam. The other end projected beyond the outer edge, and to it a horse or mule was attached. The inside diameter of the stone is slightly smaller than the outside, so that it inclines somewhat toward the centre. Sometimes, instead of having one stone only, two stones are placed on the same arm, on opposite sides of the circle, and at different distances from its centre. These wheels run over a bed of hard stone. Sometimes the crushing is done dry; but it is generally done wet—the ground ore being washed out and allowed to settle. The more modern mill* consists of a large wheel, of iron or stone, 1.65 m. in diameter and 0.38 m. wide. It is bound together with an iron tyre 0.10 m. thick. It rotates on a horizontal shaft attached to a vertical one. The other end of the shaft projects so that a mule can be harnessed to it. The wheel runs in a circular space made of iron, which is 0.50 m. wide, on the inside of which there is a screen of five or six meshes to the

* "Engineering and Mining Journal," vol. xxxiii., p. 104.

inch. When there are two wheels, the axis is generally about three metres high, and turns on pivots—one fixed in a step raised above the bottom of the grinding-space, and the other held by a frame above. The arm on which the wheels revolve is fixed to this axis, and the power is communicated by an arm fixed above on the axis. This arm may be single for one mule, or may project on both sides; in which case yokes are attached so that a mule can be harnessed at each end. The number of these mills depends on the amount of work to be done. When the amount is small, mules are always used; but when it is large, water power or steam is the motor.

In some of the works both the Chilian mill and the stamps have been abandoned for a series of crushers, *quebraderos*, or for even a single one. In this way, by a machine readily managed and repaired, a much larger amount of material can be prepared for the arrastra than by either of the other machines.

II. Grinding and Amalgamating the Ore in the Arrastra.

The crushed ore goes from the stamps or Chilian mill to the arrastra, which is a very important part of the process, as the yield of the ore depends very largely on the work which is done in it. Its action is very slow, but no machine yet invented can compete with it in the efficiency of its work. The arrastra, Fig. 104, is generally circular and somewhat below the level of the ground. It is from three to four metres in diameter. The bottom is sometimes made of the hardest boulders that can be found in the country, bedded in clay with their smooth sides turned up and ground to something like even surfaces before the operation begins. This is a bad construction, as the open places between the stones, in filling up, cause a large loss of both mercury and amalgam. It is surprising, however, that with such a rude construction the loss of mercury is not very much larger than in the better constructed arrastras. This is owing to the great skill which the men have acquired, not only in working, but in picking out the mercury and amalgam from the cracks, and refilling them with slimes. Such an arrastra will have to be run the longest time possible, fifteen or twenty days, before a clean-up is made. It will then generally be found expedient to remove the

tailings and work up all the material in the interstices. A properly constructed arrastra can, however, be cleaned up every few days without disturbing the pavement. It is generally built of paving stones or slabs of quartzose porphyry. In the best works, the edges of these stones are carefully dressed and they are put together with cement, or when that cannot be had, with the very fine tails which result from washing up the *torta*. These stones are 0.75 m. in length. They are placed vertically. When put in with care, the bottom will last for twelve months. It will then be necessary to clean out all the cracks and repair it, taking up the stones, carefully scraping them, and washing the dirt upon and that beneath them, to recover any mercury or amalgam that may have penetrated into the ground. The sides are made generally of flat stones forming a rough curbing 0.60 m. high, which projects enough to make the interior about 0.60 m. deep. In the centre of the arrastra, raised above the bottom, is a pivot hole for the central shaft, which carries sometimes two, at others four arms, and is supported above and below. To each of these arms one and sometimes two stones are attached, which act as mullers, *voladoras*, to grind the ore. They are made of quartzose porphyry, which must have an open grain so as to present a good grinding surface until it is entirely worn out. A close-grained stone would become smooth after a little wear, and would then be no longer serviceable. They are usually, when there is only one to each arm, a little smaller than the half diameter of the arrastra and about 0.40 m. thick. Two holes are drilled in each one; into these, wooden plugs are driven to receive staples, by which they are fastened to the arms by means of thongs, leather, or chains, Fig. 105, in such a way that their front edges will be about 0.05 m. above the bottom, while the rear drags. When new, all the stones together weigh from 300 to 800 kilos. The arms are sometimes niched so as to allow of changing the positions of the stones at will. There are usually four of these mullers, but sometimes only two, and in very rude arrastras, Fig. 104, only one is used. They do not last much over a month, and are sometimes worn out before that time. When they are worn down to about 200 kilos. they are replaced one at a time, so that

there are always old and new stones in the mill at the same time.

The arrangement of the arms differs according as animal or water power is to be used. When mules are used, one of the arms is made to project over the side of the arrastra, and to it one and sometimes two mules are hitched. Such arrastras are called *arrastra de mula*, or when they are of large size, *arrastra de marca*. When water power is to be used, all the horizontal arms project beyond the rim. From these arms rods descend, which support a horizontal wheel, which revolves around outside of the arrastra a few centimetres above the pit. In the circumference of this wheel, at intervals of 0.15 m., rectangular floats,

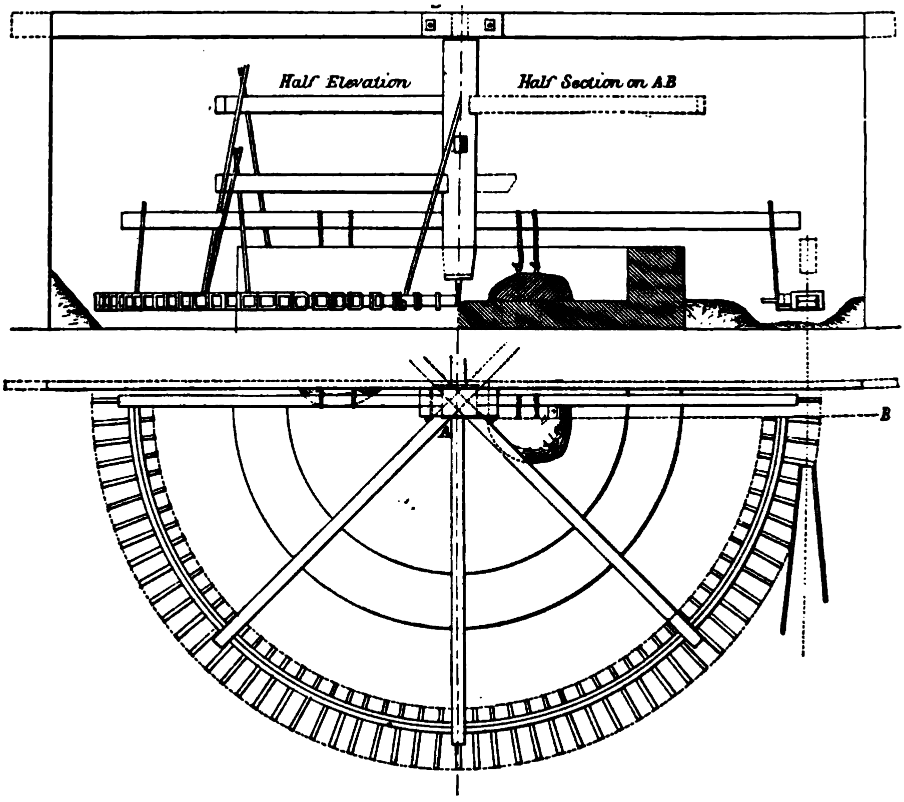


Fig. 105.

slightly concave, and set edgewise, are placed, Fig. 105. These are called spoons, *cucharas*, and these arrastras are distinguished as

spoon arrastras, *tahona* or *arrastra de cuchara*, in distinction from the *arrastra de mula.* The men in charge of the grinding are called *tahoneros.* The water strikes these paddles, the power being acquired while descending through a tapering shoot which has a fall of 0.20 m. in every 3.5 m. to 4.5 m. This horizontal water-wheel* runs in a channel a few centimetres deep on the outside of the arrastra, as shown in Fig. 105. If the central space, called *tosa*, which is the arrastra proper, is three metres in diameter, it is usually not more, and about 0.50 m. deep, the wheel six metres in diameter with a width of from 0.60 m. to 0.70 m., the outside diameter of the ditch would be about 7 m. Such an arrastra can treat between 400 to 600 kilos. of soft ore in twenty-four hours, or if it is hard, 700 and 800 kilos. in about three days. This is a wasteful appliance, but there is a superabundance of water, so that it makes little difference. These arrastras are constantly employed when water-power can be had. A wheel of this kind with a diameter of 6 metres will carry two mullers for twenty-four hours without stopping, as fast as four mules can, that cannot work for more than eight hours a day.† At Chihuahua such a wheel runs both the arrastra and the stamps. When overshot water-wheels are used, the power is transmitted by spur gearing on the upper part of the central shaft.

In some few cases an overshot water-wheel is used to run a number of arrastras. The power is transmitted by wooden gearings. When the arrastra is new, or when a new bottom has been put in, *rebajado*, it is turned either empty or with a few *cargas* of tailings, *jales*, or low grade ores, *tierras de labor*, so as to make the stones even and fill up the cracks—if the stones have been simply laid together—with material of but little value.

A good deal of importance is attached to the use of the proper quantity of water, and to the times as well as the way in which it is added. When a new bottom has been put in, one muller is attached to the arm, and it is set to work grinding up with water the residues of the washing of a *torta*, to smooth down the pavement and to fill up any cracks. This is continued for one day. The next day another muller is attached; the third day another.

* Trans. Am. Inst. of Min. Eng., vol. xi.

† Report of the U. S. Mining Commissioner, 1872, p. 436.

On the fourth day poor ores are charged; at the end of four or five days, the fourth muller is attached, and the usual work is then commenced. From one-half to two-thirds of the total quantity of ore to be treated is added at first. If there is any free gold or silver in the ore, a little mercury is added at the start in order to catch it. The quantity of gold contained in most Mexican ores is so small, that if it was not separated in some way in the treatment, it would be absorbed in the silver, and its separation by a parting process would hardly pay, so that it would be lost; but by adding mercury, especially that which has already been through the arrastra, much of it is collected. When the ores contain a very considerable quantity of native gold or silver, it is desirable to collect as much as possible with mercury in the arrastra; and if no other minerals are associated with it, the whole or the greater part of the treatment, as at Chihuahua, is comprised in its treatment here.*

The usual charge is one ton; it is often greater in large and less in small arrastras. When the charge has been introduced, a few buckets of water are thrown in to make a sufficiently consistent pulp, about half the total quantity used being added at first. If there is too little water, the ore is raised and pushed forward by the mullers without being ground. If there is too much, it packs underneath them. Care is taken to add the water as required, to keep the proper consistency. To do the work most efficiently, the mullers should be made to revolve slowly at first, but when the larger pieces have become reduced, the motion is increased to from six to ten turns a minute. This is sufficiently rapid to prevent the larger and heavier pieces from settling and thus clogging the *voladoras*, and does not make the charge rise over the sides. When the ore has been ground about eight hours, quicksilver is added in sufficient quantities to amalgamate the free gold and silver. The quicksilver is usually amalgamated with either silver, copper, or zinc. The quantity added depends on the quantity of gold and silver in the ore, and on the quantity to be worked before a clean-up is made.

When the arrastra is new, or immediately after a clean-up, from two to five kilogrammes of mercury are added at once. When

* For further details see the chapter on the Treatment of Gold in the Arrastra, in vol. ii.

the work is going on regularly, it is 0.25 kilo. every second day. If there is no free gold or silver, no mercury is added in the arrastra. When 400 kilogrammes are treated per day, which makes about 12 tons a month, 6 kilogrammes of amalgam, containing about 4.5 kilos. of quicksilver, are used. This acts readily as long as there is plenty of free mercury present; but as this becomes saturated with the precious metals, fresh quantities must be added; and to determine what this quantity should be, assays, *tentadura*, of the amalgam taken from the bottom, made by washing in a horn spoon, must be made every day or two. Sometimes the assay is made on a red earthen plate, *platillo*, which is used as a pan.

It is desirable that the amalgam should not be too liquid, for it is then liable to roll into the crevices and be caught there. If, however, it is too dry, the mercury, being already nearly saturated, will not attack the precious metal. A properly constituted amalgam flattens and spreads itself out, and presents large surfaces for contact; a liquid one rolls around in globules and may sink into the interstices; and even if it does not, is not so likely to catch the precious metal.

In some places, a quarter of the arrastra* is cleaned to the bottom, and the mixture of ore and amalgam taken out and washed. This, however, is not usually done, except in very small *tortas*, when the ores being treated are new, or for some reason do not work well. Usually the assay is taken by probing in different parts; the different probings being put together and then tested in a small vessel called a *jicara*, by pressing the thumb or finger against the side. With a very little experience the quantity of mercury is quickly arrived at without so large an assay, and the horn-spoon assay is sufficiently exact. When the amalgam is too dry, more mercury must be added. Generally, it is not desirable that the amalgam collected should contain more than 20 per cent. of gold and silver.

The quantity that a single arrastra can grind in twenty-four hours, varies with the hardness and the richness of the ore. It will generally be from 400 to 600 kilos., and will require the use of from 900 to 1200 litres of water. When no grit can be felt between

* "Engineering and Mining Journal," vol. xxxiii., p. 104.

the thumb and forefinger, the work of the arrastra is regarded as complete. When the hands of the men who do the work are not very sensitive, they sometimes make the test by rubbing some of the pulp on the lobe of the ear. There is considerable difference in the fineness of the pulp in different sections. With coarse ore, the amalgamation will not be so perfect; but those who use this practice claim that the greater yield of fine pulp does not pay the extra expense, and that the economy in production and quicker returns more than pays for the loss in yield. Those who grind fine maintain the contrary, and claim that their results are satisfactory in yield, expense, and time. Probably the differences in the qualities of the ore have led to the differences in practice in the various districts. When the assay shows that the work is properly done, water is introduced to thin down the mixture and allow the heavier particles to settle. The thin slimes, *lama*, are either dipped out into barrels and carried to the slime-pits, *lameros*, or into launders, from which they run into the settling-vats. Sometimes a spout or plug is put into the sides of the arrastra for the purpose of allowing the pulp to flow into the launders. These troughs are removed as soon as they have been used. When the pulp is dipped out, a cover is put on the floor of the arrastra to protect the amalgam. When no protection is used, care is taken not to go near the bottom. The whole of the slimes are not removed at any one time, except to make a clean-up. In the pits, the slimes are allowed to settle until they are ready to be carried to the *patio*. It takes about three days to grind a charge.

At Chihuahua, on native silver ores, the arrastra is generally charged with a ton per day of third-class ore, yielding from $250 to $1000 per ton, requiring about 25 lb. of mercury. After three days' run, ore as rich as $2500 is added, which requires more quicksilver. As much of this ore is added as is necessary for the purpose of getting a suitable amount of amalgam collected in the arrastra, preparatory to the clean-up. Some hours after adding quicksilver, the amalgamator, *azugero*, takes an assay with the horn-spoon, washes it, and judges whether the proper amount of quicksilver is present. These assays are regularly made, and by means of them great skill is rapidly acquired in learning how to add the mercury. Every morning, after the

silver seems to be amalgamated, a large quantity of water is added to the material in the arrastra, and kept in motion from four to six hours. This separates the amalgam from the fine ore, and allows the heavier particles to settle to the bottom. The fine material which has not been amalgamated runs off, carrying with it all the finely-ground ore. The coarse grains, not yet sufficiently reduced, remain and are ground in the next charge. The tails which are thus obtained at Chihuahua are poor, so poor that they are not worth more than $3 a ton for the *patio* process. They contain all the ores other than silver, except a small part of the ruby and sulphide of silver, which have been reduced at the expense of the mercury. The sulphide of silver, being ductile, is not reduced to powder, but settles to the bottom of the arrastra, and is taken out with the amalgam. Any rich tailings which come from the treatment of rich silver ore which has been added just before the clean-up, are saved for concentration or treatment.

After a number of charges have been ground, the process of grinding is stopped to allow of collecting the amalgam, which is done by scraping the inside of the arrastra with great care. This operation is called *raspando.* In the most primitive arrastras it is performed as often as twice a month, or perhaps not oftener than twice in three months. In those of the best construction it is done from two to four times a year. As a properly made pavement lasts about a year there is no necessity for doing it oftener. It is done by carefully scraping the stones and the intervals between them with a curved tool, in order to be certain to remove every particle of ore and amalgam; the amalgam so collected is called *raspa* or *raspadura.* In case the pavement is worn out, each stone is carefully scraped and washed, and the earth for a slight depth as well. In some places the *raspa* is simply washed with the addition of fresh mercury in a wooden bowl, *boliche,* Fig. 111, where most of the amalgam is collected. This operation is called *bolichar.* The tails are then washed on the *planilla,* Fig. 110, a masonry platform erected for the purpose of concentrating them. The operator here is called the *planillero.* When the ore contains gold, or in the more modern works, as at San Dimas, it is washed in a pit called a *chuza,* Fig. 106, which is also used for the treatment of concentrated tails from the *patio.*

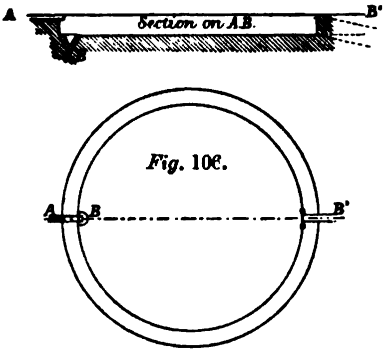


Fig. 106.

The *chuza** is an excavation 3 m. diameter and 0.5 m. deep, lined with cement, with a conical wooden bowl 0.35 m. in diameter and 0.30 m. deep, whose sides rise 0.05 m. above the cemented bottom on one side Directly above it at *A*, there is a wooden trough through which water flows freely. At the opposite end there is a trough *B*, with a gate having three plugged holes through which to let off the slimes. The scrapings are thrown into this trough and are carried by the water into the *boliche;* a boy sitting on the edge of the *chuza* keeps the material in this bowl in constant agitation with his feet, which disintegrates it. The mercury and amalgam fall into and sink to the bottom of the bowl; the heavy particles other than these are carried into the *chuza,* and the slimes run off by the trough *B*, from which, if of value, they are collected in settling tanks, and if not, run to waste. The tails in the *chuza* are concentrated by drawing out the plugs and letting the lighter material flow away; but the work is done by hand, and yields a very rich material called *cabezuela,* which is sold. When the rich tailings have been separated, the top layer of coarsely-ground ore is removed with iron scrapers and set on one side for the next charge. The amalgam is scraped up and carried in wooden bowls, *bateas,* to the washing-tanks. The gold amalgam collected in the bowl is strained and retorted as the silver is, but not with it. The surplus mercury is not mixed with that from the straining of the amalgam from the *patio.* It contains considerable gold and silver, and is always used over again to catch the free gold in the arrastra, as amalgamated mercury is always much more lively in catching free gold than pure. The amount of gold separated in this way varies from 30 to 50 per cent. of the total contents of the ore. This gives a bullion that will pay to part. The rest of the gold is recovered in the *patio,* either in the direct washing of the pulp, or in that of the *polvillos,* or is lost in the float during

* Patio Process at San Dimas; Trans. Am. Inst. Min. Eng., vol. xi.

the various processes of washing. When the ore does not contain native silver, 10 to 12 per cent. of precious metals contained are taken from the arrastra. The amalgam taken at the clean-up usually contains from 18 to 22 per cent. of silver.* The coarser the silver is, the less mercury is required.

The loss of mercury in the arrastra is owing to the formation of salts of mercury by the impurities contained in the ores, and also to the flour formed, but more especially to the latter. The losses at Guanaxuato,† where the ores contain gold, but very little native silver, it being in the form of sulphide, are given below:

Composition of amalgam	Silver ...	14 lb.	
	Mercury	56 ,,	
Weight of amalgam used		70 ,,	
Mercury added independently of amalgam			330 lb.
Mercury in amalgam			56 ,,
Gold and silver, *Plata Mixta*, obtained, including that used in amalgam	Gold ...	18 lb.	84 ,,
	Silver ...	66 ,,	
			470 ,,
Amalgam produced	Gold and silver	84 lb.	400 ,,
	Mercury ...	316 ,,	
Loss			70 lb.

As the gold was metallic it probably caused no loss. This loss of mercury is only a little more in weight than the silver contained in the bullion. It is a received opinion among the amalgamators, *azogueros*, that the loss in mercury will always be equal to the weight of silver contained in the ore.

The increase in the non-productive portion of the ore, owing to the constant wearing of the stones of the pavement and the mullers, may be as high as 8 to 10 per cent. It is a great objection to the arrastra, which has therefore been abandoned in all the other processes; but the principle of the machine is a good one, and to this principle we shall undoubtedly have to return.‡ The constant rubbing of the surfaces of the ore by the mullers, and the grinding and constant rubbing in the presence of water,

* "Engineering and Mining Journal," vol. xxxiii., p. 104 to 114.

† Phillips's "Gold and Silver," p. 333. London, 1867.

‡ I published this opinion many years ago, as the result of a careful examination of the whole subject. It has recently been brought into prominence again, Trans. Am. Inst. Min. Eng., Sept., 1885.

makes the metal bright, and the mixing brings it in contact with the mercury. It is a notable fact that in some cases in the early days of California mining, when Mexicans with their rude appliances easily made $50 to $60 a day, the most efficient modern machinery did not extract more than $15 to $20. In some instances, with the best modern appliances, an ore yielding by assay $700 to $800 did not yield more than $20 to $30 when treated in pans, while fully 75 per cent. of its value was recovered by the use of the arrastra. In ores of lower grade, the rapidity of the returns compensated for the loss, but in higher-grade ores it did not. It is a matter of great surprise that a machine has not yet been invented to work rapidly on the principle of the arrastra.

III. Treatment on the Patio.

a. Making the Torta.—The process of amalgamating in the arrastra is used when the ore contains considerable quantities of iodides, bromides, chlorides or native silver or gold. When there are none of these minerals present, it is only ground to be subsequently treated on the *patio,* as are also the tails from the treatment of the arrastra. The material from the arrastra is carried to the amalgamation court called the *patio.* This is an enclosure, more or less large, carefully paved and made as impervious to mercury as possible. It is inclined so that water will easily flow from it. Little by little, after several years' use, as the *tortas* are made over the whole surface of the court, the ground will become saturated with mercury. Every two or three years, and oftener if the pavement has to be replaced, and more especially when the *hacienda* has to be abandoned, it will be worth while to clean up and work the dirt beneath the floor. Very many methods have been tried to make and keep this flooring tight. It has been made of artificial stone, of cement, and of asphalte, and, in some places, of cut stone, faced on the edges and made tight with cement. In some places, as in Nevada and also in Mexico, timbers tongued and grooved like mill floors, and covered with water when not in use, have been laid down over an area of an acre and a half. Such a floor as this will last several years. All of these devices are excellent and work well; but as the expense is large, the old method continues

in use, and probably will do so till the whole process is abandoned, as it doubtless will be, except for very small works, in the course of a few years, when the railroads now being built are completed, and transportation becomes easy and cheap.

The slimes are called *lama;* they are brought to the *patio* as a liquid mud. In order to keep it in the place assigned for the *torta,* in small works a dam of sand or old boards is made to confine it, and it is left for some time exposed to the sun and wind, to hasten the separation of the water by evaporation as well as by drainage. In larger works, the pulp flows from the arrastra into circular walled spaces called *cajetes* or *lameros,* which are used for the same purpose. After sufficient material has been collected to treat it, and when it has acquired the consistency of thick mud, the piles, called *tortas,* or *trillas,* are made. The number and size of these depend on the size of the works. For ore they vary from 30 to 130 tons each;* for tails, they are usually smaller, or from 16 to 20 tons. They occasionally contain from half a ton to two tons; but such *tortas* indicate working on a very small scale, and the pile is trodden by men. As the material is still too liquid to support itself, a support is made around the outside with beams or stones, the joints between them being made tight with clay. Within this enclosure the pulp is placed. It will usually be about 0.30 m. in thickness. An assay is always taken both to check the work already done by the arrastra and to know what is being done. After several days' exposure, the pile will be sufficiently thick to be worked. It is spaded over and made into a regular shape of 7 to 15 metres in diameter.

b. Introducing the Reagents.—In about twenty-four hours after the shaping, from two to five per cent. of salt is scattered over the pile as evenly as possible. With ores containing from 30 oz. to 35 oz. of silver, four per cent. of salt, with those containing 45 oz. to 75 oz., about four and a half per cent. is added. The greater the amount of salt, the easier the amalgamation will be, and the more rapidly it will be effected; but notwithstanding the gain in time, it is generally found that the

* Phillips, p. 343, says that they vary at Guanaxuato from 30 to 80 montones, a montone there being 1.62 tons.

cost of the salt compensates for it, so that the amount is usually restricted to between three and four per cent. The operation of putting in the salt is called *insalmoro.* The salt which is used in the process formerly camo from the evaporation of sea-water; but this was found too expensive, on account of the long transportation. There are in Mexico a large number of salt lakes, called *lagunes,* which dry up every year. The residue contains about 20 per cent. of salt,* and fully 50 per cent. of sand. They also contain both sulphate and carbonate of soda. These impure residues, *saltierra,* are purified to be sent to the works. When purified, they contain from 70 to 90 per cent. of salt—the latter figure being seldom reached—and from 10 to 15 per cent. of carbonate of soda. The impurities make no difference in the reactions, except from there being so much less salt.

The bed of ore which is prepared with salt should be at least from 25 to 30 centimetres in thickness, depending somewhat on the consistency of the pulp. The thinner the pulp, the thicker the bed may be. In order to make the pile as homogeneous as possible, it is trodden by mules or horses—8 to 25 being required for treading a pile—the latter number being necessary for a 100-ton *torta*; 16 mules and 8 men are required for a 60-ton *torta.*† The thickness and consistency of the ore should be such that they can tread it without too much difficulty, as the work is extremely laborious. In order to have a perfectly uniform action, the slimes should not be too thick—the thickness being settled by the hoof of a mule being able to penetrate to the bottom, and to be withdrawn without difficulty—leaving a hole which does not close up for several seconds.‡ Whenever the mules stop for rest, the spading is continued. In this way the salt is thoroughly incorporated through the whole mass. This operation of treading is called *repaso.* During this time no chemical action takes place, but only a mixture of the ore and salt has been accomplished. Every possible effort has been made to do away with this treading, as it is so fatiguing to the animals, and if not well done does not allow of a full treatment of the ore. Mechanical devices of many kinds have been invented with

* Laur, "Metallurgie de l'Argent au Mexique; Annales des Mines," Series 6, vol. xx., p. 65. † *Ibid.*, p. 144. ‡ *Ibid.*, p. 141.

more or less success. Weighted wheels,* moved in various ways by mechánical devices, more or less complicated, have been tried with what seemed to be, in many cases, great success for a time; but the cost of repairs has eventually caused the return to the old way of treading with mules, which will probably be used until the process disappears. The pile is trodden and spaded during the day. The next morning it is again trodden by the mules for an hour or two, and spaded again; after which, the "*magistral*" is added. This substance was formerly a mixture of the sulphates of copper and iron, obtained exclusively by roasting iron pyrites in double-hearthed furnaces called *coma-lillos.* It contains some gangue, but this does not affect the treatment. The substance, however, is not of equal composition, as it is obtained by roasting copper pyrites of very variable yield. The following analyses of this magistral show how it may vary.

Soluble in Water.

	Poor.†	Best.‡
Water	7.60	14.84
Oxide of copper...	2.50	6.44
" iron ...	0.57	0.20
Lime	3.17	0.00
Soda	1.47	4.19
Sulphuric acid ...	9.15	9.61
Chlorine... ...	0.12	2.47
	24.58	37.75

Insoluble in Water.

	Poor.	Best.
Oxide of copper ...	5.70	0.62
" iron	20.50	23.20
" lead	0.00	7.35
Lime	7.84	0.00
Silica	38.00	28.82
Sulphur	2.22	2.80
Insoluble	74.26	62.79
Soluble	24.58	37.75
	98.84	100.54

	Poor.	Best.
Sulphate of copper	9.03	19.00
Oxide of copper	5.00	5.50
Sulphate of iron	6.75	14.80
Sesquioxide of iron	18.75	25.80
Insoluble	60.47	34.90
	100.00	100.00

In Peru,§ an ore of copper which contains as high as 13.62 per cent. of sulphate of copper, already an excellent magistral, is used for making it. This is roasted with salt, and when finished and ready to be used, contains about half the soluble sulphate

* Percy's "Silver and Gold," Part I., pp. 611 and 613.

† "Annales des Mines," 6th Series, vol. xx., pp. 75, 76.

‡ "Berg und Hüttenmännische Zeitung," 1881, p. 302.

§ *Ibid.*

that it did before. This is owing to the fact that tradition has indicated that the ore must be roasted with salt, which in this case, at least, is not only useless, but is a harmful condition. When copper ores containing sulphur are not found, but other copper ores are, these are roasted with the addition of iron pyrites for the purpose of making the sulphate of copper. When there are no ores of copper, roasted iron pyrites alone is sometimes used.* Laur cites the following experiments:

Two *tortas* of ores easily amalgamated were made and treated in exactly the same way, and at the same time. The piles were composed as given below:

	Sulphate of Copper. Torta.	Sulphate of Iron. Torta.
Dry ore	2000 kilos.	2000 kilos.
Salt	105	105
Sulphate of copper ...	6	0
,, iron		6
Mercury	12	12
Water	700	700

Each *torta* contained 2240 grammes of silver. After eighteen days, during which time it was necessary to add 16 grammes of mercury to each of the piles, each *torta* was washed separately, and the amalgam collected and distilled, with the following result:

Silver collected in the sulphate of copper torta	1890 grams.
,, ,, ,, ,, iron torta	780 ,,
Loss in silver in the sulphate of copper torta	15.6 per cent.
,, ,, ,, iron torta	65 ,,

This explains sufficiently well why sulphate of copper is preferred, although the losses in such experiments, made in a very small way, are much more than they would be in a large *torta*. But even supposing that the loss is reduced to ten per cent., with sulphate of copper used in a large way, the loss by the use of sulphate of iron would still be 41.6 per cent.

In Chili and Peru,† considerable quantities of sulphate of iron are found. It is mixed with insoluble copper ores in order to produce the necessary soluble copper salts.

$$Fe_2\,3\,S\,O_4 + 3\,Cu\,O = 3\,Cu\,SO_4, + Fe_2\,O_3.$$

* "Annales des Mines," 6th Series, vol. xx., p. 262.

† "Berg und Hüttenmännische Zeitung," 1881, p. 302.

The iron is precipitated and the sulphate of copper crystallized. The same result is obtained with malachite. Chloride of iron may also be used to produce chloride of copper.

$$Fe_2\,Cl_6+3\,Cu\,O=3\,Cu\,Cl_2+Fe_2\,O_3.$$

With atacamite, a mineral frequently found in these countries, a mixture of chloride and sulphate of copper is formed.

$$Cu\,Cl_2,\ 3\,Cu\,O+Fe_2\,3\,SO_4=Cu\,Cl_2+3\,Cu\,SO_4,\ +Fe_2\,O_3.$$

With chloride of iron the following reaction sometimes takes place—

$$Cu\,Cl_2,\ 3\,Cu\,O+Fe_2\,Cl_6=4\,Cu\,Cl_2+Fe_2\,O_3.$$

Decomposing iron pyrites can also be used—

$$2\,Fe\,SO_4+2\,Cu\,O+O=2\,Cu\,SO_4+Fe_2\,O_3.$$

This last reaction is a little slow; but if roasted pyrites are used, it takes place very quickly. If a little excess of the oxide of copper is added, no iron is left in solution. The mixture should be roasted in such a way as not to decompose any of the sulphate of copper, but all the sulphate of iron should yield its sulphuric acid to the oxide of copper; this it is almost impossible to do. But the heat makes rapid action possible.

When no sulphate of iron can be had, as in some parts of Peru, sulphate of alumina can be used:

$$Al_2\,3\,SO_4+3\,Cu\,O=3\,Cu\,SO_4+Al_2\,O_3.$$

The decomposition does not take place so rapidly or so completely as with the sulphate of iron, owing to the pasty condition of the alumina produced.

On account of the difficulty of obtaining the magistral, whose only efficiency is the amount of sulphate of copper that it contains, of the same strength at different times, sulphate of copper has been entirely substituted as magistral in many places for the roasted copper pyrites magistral, with great success and greater certainty and celerity of working; but in many places the old magistral is still used, and even when copper pyrites cannot be had, roasted iron pyrites is used.

The magistral is the most important reagent employed, and at the same time the cheapest. A little salt, more or less, makes no special difference; but an excess of magistral is always disastrous, and its effects must be attended to at once, or they will cause a

serious loss of both mercury and silver. The operation of adding the magistral is called *incorporo.* Whatever quantity of magistral is used, it is scattered evenly over the surface with the wooden shovels, and then thoroughly incorporated through the pile by digging it in, the operation being called *voltear la torta,* or turning the pile. When this has been done, another *repaso* is made, which is repeated every second or third day for about eight hours. The quantity of magistral added varies from one half to two per cent., according to the nature of the ore and the quantity of sulphate of copper contained in it; more being required as there are more sulphides. On the supposition that the sulphate of copper alone is of use, about five pounds to the ton of a 35-oz. to 60-oz. ore is required.

Generally from six to eight kilogrammes of mercury, *azogue,* is added for every kilogramme of silver contained in the ore in the *torta,* as determined by the fire assay. The amount of mercury put in at this time varies with the theory of the amalgamator. Some add two-thirds; others three-fourths of the lowest quantity at once; others add it in very small quantity at first, and the rest gradually. In any case, the effort is made to add it in the smallest globules possible, by walking over the pile and squeezing the mercury through a canvas bag containing not more than five or six kilogrammes of it, or through strainers so as to distribute it as evenly as possible over the pile.

Immediately after the addition of the quicksilver, the animals are set to treading, the spading being done when they rest. This is continued for two hours. A solution of hot sulphate of copper is then added to the pile; the quantity being larger as the ore contains sulphur, arsenic, antimony, or zinc. For ordinary pure sulphurets, about four kilogrammes to the ton are used. Precipitated copper, *precipitado,* in the proportion of one part of copper to five of sulphate, is also used. This cools the pile. After the sulphate is added, the *torta* is trodden again until 3 P.M. The mules employed for this purpose do no other work. They are generally blindfolded, and are driven in teams of not more than eight or nine. They are usually tied together four abreast, and are driven by a man who stands in the centre of the *torta* holding the halter, and who, by the aid of a long whip, makes them walk in

such a way, commencing at the outer edge, as to cover every part of the *torta*. Sometimes two teams are at work on the same *torta* when it is very large. A day's work is from 6 A.M. to 3 P.M. It is very fatiguing. When the work of treading stops, the feet of the mules are carefully washed in a tank provided especially for that purpose, not only to recover the rich material, but also to keep the mules healthy; otherwise, being in constant contact with so much mercury, they would soon become diseased. They cannot be prevented from licking themselves, however, to get the salt the mud contains. Balls of amalgam, which often weigh* from 50 to 100 grammes, are sometimes found in their stomachs; which, however, contain but little mercury.

The reactions in the *torta* commence at once after the magistral is added. It is said to work cold or hot. There are two kinds of heat: the first is due to an excess of the reagents; the second results from cold, and is called *calor de frio*. They differ as to their cause, but the result is the same, and increases the loss in mercury while it diminishes the extraction of the silver. On cold mornings, the heat of the pile being greater than that of the air, the pile steams; but as the sun rises higher this vapour ceases. This is called the *calor de frio*. When there is an excess of magistral, the chloride of mercury acts on the sulphide of silver and makes chloride of silver and sulphide of mercury—which latter is entirely lost. A large amount of heat is produced in this way. When the heat is thus caused by the excess of the reagent, very finely-ground ores containing oxide of copper, wood-ashes or lime is added to decompose the chloride of copper which is formed. Lime or ashes are, however, never added when it can be avoided; they do not revivify the mercury, and they retard the operation and diminish the yield of both gold and silver. When lime is used, it should be in fine powder, and only just enough should be added to produce the effect. If large pieces of it were used, they would not be likely to be wholly acted on by the time the *torta* was right again, and their effect would have to be counteracted, as the pile would become too cold. Tails, or any other sand free from soluble substances, can be used; but these are open to the objection that they increase the bulk

* Phillips's "Gold and Silver," p. 341.

without increasing the yield of the *torta*. When the heat is not too great, it can sometimes be cured by the application of cold water; but care must be taken not to add so much as to thin the pulp. Cold working means simply that the operation does not proceed quickly enough, and that an insufficient quantity of magistral has been added to the pile. If left in this state, a large quantity of mercury would be lost as oxide of mercury. To ascertain exactly what is to be done with the *torta* when in this state, assays of from one to three kilogrammes *ijadas*, are taken, and what is required added according to the indications which they give.

Many amalgamators prefer to work the *torta* rather hot. When it is manifestly too hot, they allow it to remain perfectly idle for a few days, taking assays all the time to ascertain when it gets back to the proper state. They add nothing to the pile in the mean time, and when it has come back to its normal condition, go on as if nothing had happened. They think that they gain time and do not lose any more quicksilver than if they worked faster, and that they get a larger yield of the precious metals. In the winter season a little less sulphate of copper is required than during the summer. They generally begin to diminish the quantity of the reagent in September.

There have been a great many theories in regard to the action of these reagents, and a great many investigations of them, which can hardly be said to have cleared up many of the obscure points. A résumé of what has been done is given below, which, however, is not very satisfactory, and does not throw much light on the subject. Some of the published reactions, after careful trial, could not be obtained. The reactions given below have been compiled in the hope that some one may be led to make a more careful examination of the whole subject.

The amalgamators suppose that the chloride of sodium cleans the silver and the sulphate of copper heats it, and that the amalgam of silver and mercury results. The mercury lost is counted as lost mechanically; the amount of loss being about equal in weight to that of the silver extracted. The generally received theory is, that the salt and the sulphate of copper act, the one on the other, and give rise to chloride of copper and sulphate of soda:*

$$Cu\,SO_4 + 2\,Na\,Cl = Cu\,Cl_2 + Na_2\,SO_4.$$

* "Berg und Hüttenmännische Zeitung," 1881, p. 303.

The chloride of copper acts on the metallic silver and the sulphide of silver; chlorides of silver are formed, which are dissolved in the excess of chloride of sodium.

$$2\ Ag + 2\ Cu\ Cl_2 = 2\ Ag\ Cl + Cu_2\ Cl_2.$$
$$Ag_2\ S + Cu\ Cl_2 = 2\ Ag\ Cl + Cu\ S.$$

When mercury acts on artificially prepared chloride of silver, it reduces it to a metallic state, when it enters into combination with the mercury.

$$2\ Ag\ Cl + 2\ Hg = Hg\ Cl_2 + Ag_2\ Hg.$$

This reaction takes some time, and is less sensible on the natural than on the artificial substance.

If chloride of copper is treated with mercury, sub-chloride of copper and sub-chloride of mercury are formed.

$$2\ Cu\ Cl_2 + 2\ Hg = Cu_2\ Cl_2 + Hg_2\ Cl_2.$$

This reaction takes place more rapidly than with chloride of silver. If chloride of iron is substituted for the chloride of copper, all the reactions take place, but much more slowly, and this is especially true when sulphide of silver is present. The presence of salt accelerates the reactions in all cases. If any of the metallic silver in the ore has not been transformed into chloride, this is attacked directly by the mercury.

When sulphide of silver and mercury are shaken together, sulphide of mercury and amalgam are formed.

$$Ag_2\ S + 2\ Hg = Ag_2\ Hg + Hg\ S.$$

This reaction is slow, but much quicker than with chloride of silver. All the sulphide of mercury is entirely lost.

Rammelsberg and Huntington have recently made the following investigations.* If sulphide of silver and chloride of copper are made to act on each other, either sub-chloride of copper, chloride of silver and sulphur are produced, or the sub-chloride of copper formed becomes a sulphide.

$$Ag_2\ S + 2\ Cu\ Cl_2 = 2\ Ag\ Cl + Cu_2\ Cl_2 + S.$$
$$Ag_2\ S + Cu_2\ Cl_2 = 2\ Ag\ Cl + Cu_2\ S.$$

The liberation of the sulphur is, however, a secondary reaction, taking place only to a very limited extent, thus:

$$Ag_2\ S + Cu\ Cl_2 = 2\ Ag\ Cl + Cu\ S$$
$$Cu\ S + Cu\ Cl_2 = Cu_2\ Cl_2 + S.$$

* "Die Metallurgie des Silbers und Goldes," von. J. Percy, p. 12, Brunswick, 1881, and "Engineering and Mining Journal," vol. xxxiv., p. 150.

When the solution is boiled for some time, the sulphur disappears and sulphuric acid is formed. The amount of sub-chloride formed, and of sulphur set free, is dependent on the strength of the solvent, which in this case is salt, on the temperature, and on the presence of air. The secondary reaction depends on the power of the solution to dissolve the chloride. If this could be removed, the solvent power of the solution would be to a certain extent regained. The action of the air in facilitating the secondary reaction is due to its converting the sub-chloride into an insoluble oxy-chloride.

$$3\ Cu_2Cl_2 + 3\ H_2O + 3\ O = 3\ CuO,\ Cu\ Cl_2,\ 3\ H_2O + 2\ Cu\ Cl_2.$$

If chloride of copper and sulphide of silver are boiled together the decomposition is complete.

$$Ag_2\ S + Cu\ Cl_2 = 2\ Ag\ Cl + Cu\ S.$$

When sub-chloride of copper and sulphide of silver are mixed, the following reaction takes place:

$$Ag_2\ S + Cu_2\ Cl_2 = 2\ Ag\ Cl + Cu_2\ S.$$

When one hundred parts of the sulphide of silver are treated with sub-chloride, in a solution of salt, as much as 7.6 or 8.3 per cent of the silver remains dissolved in the salt solution. When the residue was treated with zinc, the following reaction took place:

$$Ag_2\ S + Cu_2\ Cl_2 + Zn = Zn\ Cl_2 + Cu_2\ S + 2\ Ag.$$

When a salted solution of sub-chloride of copper is mixed with a saturated solution of chloride of silver in salt, no precipitation takes place, nor can it reduce chloride of silver when it is in powder. If sulphide of silver is added to the salt solution of sub-chloride of copper, chloride of copper, sulphide of copper, and metallic silver are produced.

$$Na\ Cl,\ Cu_2\ Cl_2 + Ag_2\ S = Na\ Cl,\ Cu\ Cl_2 + Cu\ S + 2\ Ag.$$

It thus appears, that while it cannot affect the chloride of silver, the sub-chloride of copper can reduce sulphide of silver, which, in the presence of mercury, is amalgamated without having passed into the state of chloride at all. If ammonia is added to the solution of the sub-chloride of copper and chloride of silver, silver is precipitated.

$$2\ Ag\ Cl + Cu_2\ Cl_2 + N\ H_3 = 2\ Ag + 2\ Cu\ Cl_2 + N\ H_3.$$

When chloride of silver, sulphide of copper, and ammonia are heated, a blue solution is obtained. One half the chloride of silver is converted into sulphide of silver. The residue, which is black, is composed of sulphide and chloride of silver, and contains no copper.

$$8\ Ag\ Cl + 2\ Cu\ S = \underbrace{Ag\ Cl + 2\ Ag_2\ S}_{\textit{Not Dissolved.}} + \underbrace{3\ Ag\ Cl + 2\ Cu\ Cl_2}_{\textit{Dissolved.}}$$

Three parts of chloride of silver and two of chloride of copper remain in solution.

If two parts of chloride of silver dissolved in ammonia are treated with sub-sulphide of copper, a mixture of silver and sulphide of copper is precipitated, about one-tenth of the silver still remaining in solution.

$$2\ Ag\ Cl + Cu_2\ S = 2\ Ag + Cu\ S + Cu\ Cl_2.$$

If four parts of chloride of silver are used, the copper remains almost entirely in solution, and 28.2 parts of the silver are also in solution. The residue consists of metallic silver and sulphide of silver.

$$4\ Ag\ Cl + Cu_2\ S = 2\ Ag + Ag_2\ S + 2\ Cu\ Cl_2.$$

Professor Huntington found that when chloride of silver and sulphide of copper are mixed in an ammoniacal solution, sub-chloride of copper is formed, which, reacting on the chloride of silver, forms metallic silver and chloride of copper.

$$2\ Ag\ Cl + Cu_2\ S + NH_3 + Ag_2\ S + Cu_2\ Cl_2\ N\ H_3.$$

$$2\ Ag\ Cl + Cu_2\ Cl_2\ NH_3 = 2\ Ag + 2\ Cu\ Cl_2\ N\ H_3.$$

The chloride solution for this reaction must be kept at a certain strength or the reaction will cease, and anything which causes further dilution will undo a part of the work already accomplished.

When chloride of copper and sulphide of arsenic are mixed, rapid decomposition takes place, and a precipitate of sulphide of copper and chloride of arsenic is formed.

$$As_2\ S_3 + 3\ Cu\ Cl_2 = 3\ Cu\ S + 2\ As\ Cl_3.$$

When chloride of copper and sulphide of antimony are mixed, a precipitate containing sulphur, copper, oxygen, chlorine and antimony is formed. Some antimony remains in solution on account of the sulphuric acid formed. When sub-chloride of

copper is used, most of the copper is precipitated in the metallic state.

If proustite and pyrargyrite are treated with chloride of copper both are decomposed. All the silver of the pyrargyrite is converted into chloride, while only a part of that in the proustite is so acted on. The reaction for proustite is

$$4\ Ag_3\ As\ S_3 + 12\ Cu\ Cl_2 = 8\ Ag\ Cl + 2\ Ag_2\ S + 2\ Cu\ S + As_2\ S_3 + 5\ Cu_2\ Cl_2 + 2\ As\ Cl_3 + 5\ S.$$

and for pyrargyrite

$$2\ Ag_3\ Sb\ S_3 + 4\ Cu\ Cl_2 = 6\ Ag\ Cl + Cu_2\ Cl_2 + 2\ Cu\ S + Sb_2\ S_3 + S.$$

In both cases part of the reagents remain in solution.

When sub-chloride of copper is dissolved in salt and boiled with pyrargyrite in powder, a black product is formed which contains most of the silver, all the antimony and sulphur, and some copper and chlorine; 7.3 per cent. of the silver is dissolved by the salt.

$$2\ Ag_3\ Sb\ S_3 + Cu_2\ Cl_2 = 2\ Ag\ Cl + Ag_2\ S + 2\ Ag + 2\ Cu\ S + Sb_2\ S_3.$$

When sub-chloride of copper and proustite are treated together, a grey product is formed which contains nearly all of the silver and sulphur, two-thirds of the arsenic, and considerable portions of the copper and chlorine.

$$6\ Ag_3\ As\ S_3 + 7\ Cu_2\ Cl_2 = 14\ Ag + 4\ Ag\ Cl + 12\ Cu\ S + 2\ As_2\ S_3 + 2\ Cu\ Cl_2 + 2\ As\ Cl_3.$$

The proportion of the silver dissolved in the salt was 4.7 per cent.

If mercury, sulphide of silver, chloride of sodium, sand, and water are worked together, seven-eighths of the silver present is extracted, three times as much as when the salt was not there. If oxide of iron is present in the mixture, chloride of iron will be formed, which is reduced to sub-chloride by the mercury, and a chloride of mercury is formed. A very small amount of oxide of iron produces a very considerable loss, as the sub-chloride constantly changes to chloride of iron, in contact with the air. If to this last mixture sulphate of copper is added, a little less silver is obtained, and the loss of mercury is large. If proustite, which contains 65.5 per cent. of silver, 15.1 of arsenic, and 19.4 of sulphur, is substituted for the sulphide of silver, twice as much silver combines with the mercury when chloride of copper is

present. It requires a great deal of shaking to decompose the sulphide of silver. When sulphide of zinc is present with chloride of copper, it causes the formation of sulphide of copper and chloride of zinc, so that ores which contain blende always amalgamate badly.

From these reactions it would seem that, during the first two or three days, chlorides of copper and iron are produced by the action of the magistral on the salt; that chloride of silver is formed by the action of these chlorides on the easily attacked ores, and even on the sulphide of silver; that the chloride of silver is dissolved probably at once in the excess of salt. In chlorurising the silver, the copper and iron salts have become reduced to chlorides, which in the presence of sulphide of silver form the chlorides, and produce metallic silver, or, when it is absent, quickly become oxy-chloride, and produce no further action. Sub-chloride of copper reduces sulphide of silver; but the sub-chloride of iron does not. The presence of mercury prevents the formation of an excess of chloride of copper, for as soon as there is an accumulation of it, it acts on the mercury and is reduced to sub-chloride. Just as soon as the mercury is introduced the free silver is amalgamated; the chloride of copper which is still in the pulp forms calomel and sub-chloride of copper, which acts on the sulphide of silver and leaves it as metal, to be acted on by the quicksilver.

Some authors, especially Mr. Bowering, deny that chloride of silver is formed at all, as none was found in a *torta* for four months on the *patio*, during which time he constantly examined the piles. Mr. Bowering says, in support of this theory, that when only two of the re-agents, sulphide of silver, chloride of sodium, or sulphate of copper are mixed together, no effect is produced, and that when three are mixed in a small vessel, the mercury combined with just half of the chlorine in the chloride of copper, and formed sub-chlorides of both metals. As the chloride of copper has the property of absorbing oxygen, he concludes that it is the principal reagent. According to this theory the mercury acting on the chloride of copper makes sub-chlorides of both. The chloride of copper absorbs oxygen, which acts on the sulphide of silver and makes sulphuric acid, and leaves the

silver in a metallic state to be absorbed by the mercury. The sulphuric acid set free acts on the chloride of sodium, and forms sulphate of soda. Chlorine is given off, combines with the sub-chloride to make a chloride of copper, which is again decomposed, and so on. In this case the sub-chloride acts just as nitric acid does in the manufacture of sulphuric acid. The action of the chemicals in the pile is especially slow if sulphide of silver is present, in which case the loss of mercury is also very large. When the whole of the silver is in the state of sulphide, a large part of it, which may sometimes be as high as 40 per cent., is lost. The mercury transforms the chloride of copper into sub-chloride, which, like chloride of silver, is soluble in an excess of salt. The sub-chloride in this state acts more energetically on the sulphide of silver than the chloride. A sulphide of copper is formed, while the silver is precipitated, and the chloride of copper formed again by giving up half the copper, which becomes a sulphide. This advantage is gained only at the expense of a very large quantity of mercury; and in order to prevent this loss, experiments were made of not introducing the mercury until much later in the process, but this did not succeed, as the extraction of the silver was not so well done.

The next day after the first treading, another one is made. The torta is then allowed to rest for a day, with occasional spadings, quite as much to make the mixture as to ascertain whether the ore is not getting too stiff from evaporation. As the heat of the sun is depended on for a part of the chemical action, water, when added, must be added in the morning, so as not to cool down the *torta* after it has once become heated, and thus disturb the reactions which are taking place. The pile must be trodden several times, the object being to keep renewing the surface of the silver which, without this, would become rapidly covered with a bed of solid amalgam which would prevent further action. The operation lasts from three to six weeks, according to the way in which it is conducted, the temperature of the air, and the size of the heap. A succession of cloudy days or cold weather in the summer time will retard the operation. Continued or heavy rains may so thin the pulp as to prevent the reactions taking place, and stop all the working until the pulp

thickens up again from evaporation. When all the conditions are the most favourable, the *incorporo* can be completed in 15 to 18 days. When they are unfavourable, it may take from 40 to 50 days. Taking several months together, 20 to 25 days will be the average time. In winter, when the *torta* always works slow, it may last as long as two or three months.

The day after the mercury is added, assays, *tentaduras*, are made, to see how the *torta* is working, to learn if any one of the reagents used is required, or if any of them is in excess. To do this, a probe-sample, which will weigh about 250 grammes, is taken from as many different parts of the pile as possible. The assay is washed in a horn spoon or in an earthen plate, *platillo*, 0.18 m. in diameter and 0.02 m. deep, a rotating motion being given to it. The lighter particles are carried off, and the heavier ones deposited on the bottom in the order of their gravity—the heaviest being in the centre. The mercury which has not yet acted, is generally in the centre, the silver-white amalgam, *ceja*, which, when moved, shows a distinct tail, *lista*, next to this, then the undecomposed black sulphurets, then pyrites, and generally a fifth ring of mercury in flour. Three assays are generally made on the *torta* each day, one in the morning before the work commences, one after the treading is about half done, and a third after it has been completed. During the first few days, the appearance of the mercury remaining unacted upon shows the workmen what is taking place. The mercury is always more or less attacked. If during the first day it looks dull, is of a deep grey or lead colour, there is too much magistral, and the *torta* is said to be too hot, and the temperature is really too high. A little lime is then added which decomposes part of the sulphate of copper and slackens the action. Lime is sometimes replaced by alkaline ashes. If on the contrary, the mercury is perfectly brilliant and not acted on at all, or is broken up into little globules, or if it is of a slightly yellow tinge, the *torta* is too cold, and more magistral must be added. It is always better to have too little than too much magistral; more can always be added, but too much means a loss of mercury. When the amalgam, *limadura de plata*, is in the proper condition, it is in thin scales, which are easily collected together into a mass of dry silver amalgam, *pasilla*, and

mercury is easily pressed from it with the fingers. When it is very thin, so that it easily breaks up into fine globules, it is said to be *debil*, or weak. When it is hard and crystalline, and so dry that no mercury comes out from it when it is pressed, the amalgam is said to be strong, *fuerte*, and more mercury must be added. A dirty blackish appearance to either the mercury or amalgam indicates improper working. When the indications of colour are all right, but the assay shows that no progress is being made, salt must usually be added. Sometimes this condition is only temporary, and is owing to a sudden reduction in the temperature of the air. Generally the defects are owing either to heat or to cold. Excessive heat always signifies a loss in mercury, and should be stopped as quickly as possible by adding cold water or ashes. If the heat is not excessive the *torta* may be allowed to stand a few days. Cold working is remedied by the addition of salt, or of sulphate of copper, or by additional treading. To ascertain which of these is required, careful assays must be made. Generally in the commencement fresh ore or cement copper is used to correct the working, and toward the close cement copper ashes, or lime.

When the amalgam is very fluid and easily breaks up into very small globules, and the assay shows that at least 75 per cent. of the silver in the pile is amalgamated, the *torta* is said to be finished or *rendida*. Sometimes the assay shows everything to be right, but no progress is made for several days in the amalgamation. This is usually owing to a want of salt, or to cold. If, on examining the black sulphurets, *polvillos*, and rubbing the small metallic globules of mercury or amalgam found among them with the finger, they unite to a large globule, the pile is nearly finished. If they yield a dry amalgam, it is not. The best way to ascertain this, is to make a fire assay of the original pulp and of the *torta*, and to judge by the yield. When the ores contain galena and blende, these substances decompose the chloride of copper, and the sulphur goes to the copper. The proportion of magistral to be added must, therefore, be largely increased, notwithstanding the fact that the loss in silver is always greater when there is an excess.

When the amalgamation is complete, a considerable quantity of

mercury, in addition to that required for the amalgamation of the silver, is added, with the object of making sure the collection of all the mercury and amalgam. In some districts this additional mercury is called *baño*. The pile is still trodden for some time. This last addition of mercury has for its effect to make the amalgam a little more fluid, so that it may be collected more easily, and to collect the floured mercury which would not be caught in the subsequent washing, and to prevent as far as possible further action of the reagents on the amalgam.

There is always a loss of mercury equal in weight to that of the silver contained in the ore. A further loss of from 7 to 10 per cent. comes from that which is mechanically carried off either in the *patio* or in the washing. With such ores 40 per cent. of the silver is often lost. The loss of mercury is often from 100 to 200 per cent. of the weight of the silver obtained. As a mean it is from 140 to 160 per cent., or seven or eight times the loss in the Freiberg barrel amalgamation process. The attempt was made to diminish this loss by adding a little iron, but in order that the effect may be sensibly felt, a large amount must be used, which increases the expense and does not diminish the loss much. In some of the works the mercury is replaced by an amalgam containing 30 per cent. of copper, which reduces the loss materially. The effect of the copper is the same as that of the balls of copper or iron which are used in the Freiberg barrel amalgamation process. Too much copper, however, must not be added, or it would make the amalgam of silver too friable. The loss in silver is increased by this method, but the loss in mercury is reduced to 120 to 150 per cent. The attempt was also made to use a lead or tin amalgam, but this was too viscous, and became easily reduced to powder, so that the loss in silver was increased.

At the end of a time, more or less long, no mercury is found. The operation is nevertheless continued until the amalgam attains a certain consistency. If, however, the amalgam becomes too thick, a fresh charge of mercury must be made, adding it little by little. Sometimes the assays are made over only a small part of the *torta*. A little salt will be added in one part and a little magistral in another. Assays are then made to see the effect, in order to show what should be done with the whole pile.

When the *torta* is *rendida,* it must be washed as soon as possible. If allowed to stand, the sulphur and the sulphate of copper which have not been decomposed commence to act, and cause a considerable loss of silver in the state of very finely divided amalgam, *desecho,* which will not unite. It is to prevent this as much as possible, that the large excess of mercury is added, but nothwithstanding the excess of mercury the pile must be washed at the earliest possible moment.

c. Separating the Amalgam.—The amalgam, with the excess of mercury, is scattered through a large mass of pulp, from which it must be separated by washing. This should be done once in twenty-four hours.

The washing, *lava,* is done in a box settler, *lavadero,* or in a tub, *tina*; both of these methods being in use in different works. The former is by far the most ancient. The tub, which is very much like the dolly tub or settler of California, has been in operation for many years, but as it requires the use of power, is only adopted in the large haciendas.

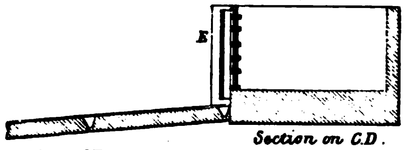


Fig. 107.

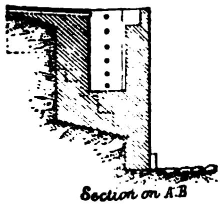
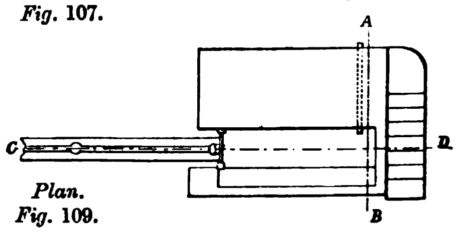


Fig. 108.



Plan.
Fig. 109.

The box, *lavadero,* Figs. 107 to 109 is built of stone on the sides, and lined with cement. It is two metres long, half a metre wide, and one metre deep. It has a platform on one side on which to pile the material to be treated. The front is closed with plank in which there are six holes, 0.05 m. in diameter, five of which are closed with plugs of wood. These serve to let off the slimes. In front of these holes is a vertical wooden trough which carries the slime to an incline trough, the bottom of which is

provided with several mercury traps to catch any mercury or amalgam that may be carried off.

The *lavadero* is built directly against the *patio*, the pavement coming up to its front wall. The material from the *torta* is carried to the platform of the settler by a pair of steps built on the platform side. The box is first filled half full with water. Two men then get into it, while one man on the platform shovels the ore into it. The men dance in the water, keeping it in motion with their feet, but keeping their hands out of it. The pulp and water are added little by little, the pulp by a single spadeful at a time, until the slimes flow out of the top hole, while the water is allowed to flow in only as fast as it flows out. The discharge falls down the vertical to the inclined trough, over the mercury traps and riffles, and goes from there to the settling tanks. The heavier liquid below is from time to time discharged by removing the lower plugs. The men are obliged to use a great deal of discretion at this work. If they work too fast, there is danger that some of the amalgam and mercury will be carried off. If they work too slow, the heavier particles collect at the bottom, and the small particles of amalgam sink through it slowly or not at all. They know by experience from the difficulty of moving their feet, when it is time to discharge through the lower holes. They never allow the lower part of the box to become filled. The amalgam is not removed until after the whole of the *torta* has been washed, then the supply of water being kept up, the plugs are removed one by one and the amalgam collected. The *lavadero* costs but little to build, but requires the labour of six men, treading, charging and bringing the pulp. As capital is scarce, but labour very abundant, the use of this settler is almost universal in Mexico.

In some districts where wood is cheap, the tub, *tina*, is substituted for the stone box. The agitation in this is done with shovels or poles from the sides. No better results are obtained, but the labour is less severe. These box settlers can only be worked during the day, and must, on account of the danger that some one else may remove a part of the amalgam, be cleaned up every night. When the washing of the *torta* is done in wooden tub settlers, *tinas*, they are usually driven by water power. They

are from two to five metres in diameter, and one and a half to three metres deep. The shaft carries four arms, which are fitted with pieces of wood 0.06 m. square and 0.10 m. apart, which reach to within 0.30 m., or less, of the bottom of the tub. In the sides of the tub there are two holes, one 0.8 m. from the bottom, which is 0.15 m. in diameter, from which the tub is emptied; the other, 0.25 m. from the bottom, is 0.02 m. in diameter, and from it the water overflows, and the tail assays are taken. The axis is geared by wooden gearing to a water-wheel. These tubs were formerly constructed of stone. Three of them communicating with each other, were placed together, and were connected by one large wheel driven by two mules trained especially for the purpose. The first of these tanks, into which the pulp was put, was called *tina cargadora*, the third, from which the discharge was made, was called *discargadora*, or discharge tank. In some of the works these tanks are disconnected, though driven by the same power, each tank being used by itself. The tank is filled one-third full of water, and the axis is set in motion quite rapidly; when mules were used they were set at a full gallop, and a charge of 300 kilos. thrown in. Water is added until it reaches nearly to the top of the tub, and the speed reduced until it is just sufficient to keep the pulp off the bottom. In about an hour the assays taken from the top hole show that the mercury has all settled. The bottom plug is then removed, and the contents of the tub discharged into the settling tanks. This is a much better and quicker method of working. There is no danger of the tub becoming clogged at the bottom, and there is no necessity for constantly cleaning up at very short intervals. The tub can be kept going night and day until the whole *torta* is washed, without any danger of having a clean-up made by others. It will undoubtedly take the place of the box settler wherever there is sufficient capital to erect it.

When the whole *torta* has been washed, the *patio* must be carefully scraped, and also the interstices between the stones, to remove any particles of pulp, amalgam, or mercury. All these scrapings, *raspadura*, are mixed with the last of the pulp, and are thrown into the settler. The time required to work depends on the number of settlers. It is usually not less than two or three days.

The tails from the *lavadero* or *tinas* consist mostly of iron pyrites mixed with the black sulphides and some ore, their proportion being different with the different ores treated. They are called *cabezilla* or *cabezuela.* They contain some amalgam which is recovered. Formerly* they were carried in wooden *bateas* to a tank filled with water, called the *pila apuradora.* On its surface a wooden bowl, *batea apuradora,* floats, which is from 1 m. to 1.50 m. in diameter. The man who washes with this *batea* leans on the side of the *pila,* and, taking hold of the bowl with both hands, gives it a peculiar motion, taking up a small quantity of water, which, after going round the *batea,* is discharged, taking some of the *cabezilla* with it. The residues are treated on the *patio.* Generally the tails from the *tinas* and *lavadero* are run over riffled launders, where some of the mercury and amalgam is caught, into two tanks connected with each other, which for a *torta* of 12 to 15 tons are five metres long, three wide, and one deep. These are called the *tanque* and *contratanque.* The object of the first is to catch all the heavy materials, such as the amalgam and the coarse particles of pulp. Most of the material containing silver and gold is caught here. The *contratanque* catches only the lighter particles, which are much poorer, and are always kept separate, unless found by assay to be of approximately the same value. The tails from the *contratanque* run to waste. The materials caught in both tanks are concentrated in the *chuza,* Fig. 106. Some amalgam, generally not less than 15 kilos., is caught here, the amount varying with the care that has been taken in the washing.

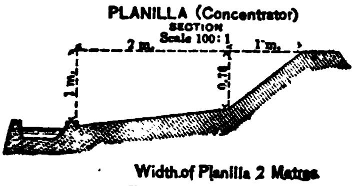


Fig. 110.

The tails from here are concentrated on the *planilla,*† Fig. 110, which is a platform of masonry from one and a half to two metres wide, and with a slope of one metre in ten towards the trough, which supplies a small stream of slowly-running water.‡ The wall at the upper side is sloped, and furnishes a space to pile up

* Phillips's "Gold and Silver," p. 344.

† Patio Process at St. Dimas. Trans. Am. Inst. Min. Eng., Vol. 11.

‡ "Annales des Mines," Series 6, vol. xx., Plate II., Figs. 5 and 6.

the tails to be washed. The workman, *planillero,* sits on a strip of board put across the water trough, and with a horn spoon containing about a quarter of a litre throws the water upon the pile of tails. The operation is commenced at the lower left-hand corner, and continued across the *planilla,* going back again when the *planilla* has been crossed to the lower left-hand corner, and working always in the same direction. The water is thrown in such a way as to spread out as much as possible, but not to splash. When this has been repeated several times, the sand for about one metre from the water-trough is thrown away. The heavy particles are thrown up on the pile, and the operation recommenced. When the supply of tails is exhausted others are added. The result of the washing is a small heap of black sulphurets, called *polvillo.*

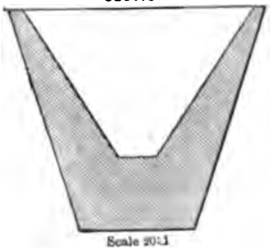


Fig. 111.

These are further concentrated in a wooden bowl called a *boliche,* Fig. 111, which has the shape of an inverted truncated cone 0.62 m. in diameter and 0.4 m. deep, which is a hand device in every way similar to the keeve used in dressing copper ores on Lake Superior,* and in the concentration of gold ores in California.

The *boliche* is sometimes made as deep as 0.8 m., and correspondingly large, though this is not usual. Water is put into the *boliche,* and the sulphurets added and stirred, and then allowed to settle. During the settling it is tapped on the outside with a stick or mallet. The heavy particles containing the sulphurets settle to the bottom, and the sand is on the top. The water is soaked off with rags. The sand is scraped off and thrown away. Below is a brownish layer of poor sulphurets called *colas,* which are removed to be roasted. Below them are the clean *polvillo* and a small quantity of amalgam. The *polvillo* is sent to Europe with the high grade ores for treatment. The roasting of the *colas* is done in an ordinary pile with a central chimney. It is put in layers 0.25 m. thick, and is used damp in order to be able to manage it better. The cover is made of earth. The pile is set

* "Metallurgical Review," vol. ii., p. 400.

on fire, and when the roasting is completed the half-burned sticks are removed. Only a part of the material is properly roasted, but it is all ground in an arrastra and added to the *torta*. In some places where fuel is cheap, the roasting is done in a reverberatory furnace, and is consequently much better done. There is always great uncertainty in roasting in piles. This roasted material was formerly treated in a *torta* by itself; but it consumes a great deal of mercury, and does not give very satisfactory results. It is much better to mix it in the piles with the ore.

In some places, during the washing, a product is obtained which contains gold and silver, and though there is not much of it, it is richer in gold and silver than the original ores, and also contains some little amalgam. As the material is mostly pyrites, it is concentrated, ground, and roasted, and used as a magistral Sometimes 2 per cent. is obtained in this way.* Of late years, in some *haciendas*, all the tails of the different operations have been treated by the Von Patera process.

IV. Treatment of the Amalgam.

The liquid amalgam is carefully removed from the bottom of the settler. All that which is caught in the mercury traps is added to it. This is carried to the mercury house, *azogueria*, and put into a large trough, originally always of stone, but now often made of iron. When the whole has been collected, a large amount of mercury, usually ten to fifteen per cent. of the quantity of quicksilver used in the arrastra, is added to the amalgam in order to clean it. It is covered with water to prevent splashing, and carefully worked over. Whatever impurities rise to the surface are removed with a cloth, and fresh water is again added. This operation is repeated until the surface becomes and remains bright. The amalgam is dried and weighed, and is then put into a conical canvas bag, like those used in the West, which is called *manga*, set over a receptacle made of hide, *pila*, to catch the drippings, which, as they contain some little silver, are of more value in the next charge than pure mercury. This is put into flasks for preservation. The amalgam, free from everything

* "Engineeting and Mining Journal," vol. xxxiii., p. 104.

except mercury, *copella*, after hanging for several hours, is ready for retorting.

At Chihuahua,* where very rich ores of native silver are treated, the amalgam looks like a coarse sand, but by the addition of mercury the dirt is removed from it. This dirt, however, is very rich, and is further concentrated. When particularly pure silver is required, it is carefully washed, and ground on a stone, in order to remove the sulphide of silver; the result is a very pure amalgam, which yields silver purer than fine bars. The amalgam cleaned with mercury is strained in canvas cloths, and the excess of quicksilver pressed out. It is made into small balls 0.05 m. to 0.06 m. in diameter, by rubbing them with the hands. This is the only way they have been able to get very high-grade silver.

Formerly, all the amalgam was beaten and pressed into an iron mould, to make bricks of amalgam, *bollos*, of such a shape that when six were placed together they formed a circular cake with a round hole in the centre. One ton of these was piled on iron supports, over a stone tank filled with water to nearly the top of a copper or iron bell, *capellina*,† which is 0.90 m. high, and 0.45 m. in diameter. This left a space 0.02 m. between the amalgam and the bell which was lowered to its place by pulleys. A wall of *adobes*, leaving a space 0.20 m. between the bell and the wall, was then built around it and fired with charcoal for fifteen hours, and removed when cold. The yield of silver was about 200 kilos., and the charcoal used about 250 kilos. per charge. This process is now abandoned.

In the more modern method, the strained amalgam is charged into quicksilver flasks from which the bottom has been removed, Figs. 112 and 113. Into these flasks others, open at both ends, are fitted so that the lower parts are beneath the surface of the water, in a tank placed under the furnace. The two flasks are luted so that the quicksilver has no outlet except into the water, where it condenses, as the screw in the upper part of the upper flask has been firmly set. The inside of the flask is then washed with milk of lime or lined with brown paper to prevent the silver from adhering

* Mining Commissioners' Report, 1872, p. 437.

† "Annales des Mines," Series 6, vol. xx., Pl. II., Fig. 2.

to the sides. To make sure that the amalgam fills the whole flask, it is first rammed in, and then pounded down with a heavy mallet.

Fig. 112.

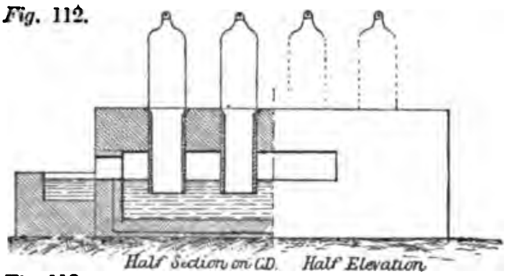
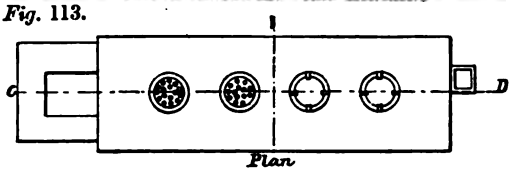


Fig. 113.



30 to 35 kilos. are charged in each of the flasks, which are then set aside to drain off the excess of quicksilver, and to allow the amalgam to harden. The amalgam is kept in place by four narrow strips of iron set into the mouth of the flask, and bent so as to cross it beneath the amalgam. As soon as three to four flasks are ready, they are taken to the retorting furnace, *quemadero,* Figs. 112 and 113, where they are set on end over holes on the slab which forms the top of the furnace. This slab is 0.60 m. above the ground. The space between the upper and lower flasks is covered with an iron plate full of holes 0.005 m. in diameter, which is luted to both the upper and lower flask. There is no danger of the amalgam falling out, except with an improperly-managed fire. The size of the furnace is very variable, depending on the amount to be treated. It may have places for eight to ten retorts. In Mexico it has entirely superseded the old *capellina.*

A wash of clay, about 0.005 m. thick, is put around the upper flasks to protect them from the air. A brick wall laid up temporarily is then built round all the flasks, and a charcoal fire is made inside of it. The first object of the fire is to dry the clay coating; it is therefore made to burn very slowly at first, so as to make it dry without cracking. When this has been done a brisk fire is made over the whole of the flasks. Water is kept con-

stantly flowing into the tank below the flasks, both to keep up the supply and to keep it cool. The mercury driven out of the amalgam falls into the water and collects there.

The operation needs care. If the temperature is raised too high, there is danger of melting the amalgam. If raised too quickly, there is danger of explosion from the rapid formation of the vapours of quicksilver. If the heat is not high enough, the bullion is impure from excess of mercury. As a precaution against this, the purchaser has a right to heat the bars of silver red hot to drive off any excess of mercury, but if the bars melt while undergoing the process, the purchaser pays for them at their weight before heating. If they do not, the weight after heating is accepted. When the work is properly done, the silver still contains one per cent. of mercury. The mercury collected in the tank is not entirely free from silver, and must be strained. The amalgam collected is called *estrujon,* is much drier than the other amalgam, and is retorted by itself when enough has been collected to make it worth while to do so.

The retort silver, *plata pasta,* is refined in a small reverberatory furnace built of *adobes,* and heated with wood, which receives a charge of 300 kilos. of the crude bullion. This charge is refined in four hours. A little litharge and lead are added to remove the impurities, which are generally sulphur, arsenic, lead, iron, and sometimes zinc. Borax and carbonate of soda are used as a flux. The loss is seven per cent. of the crude bullion, and consists mostly of quicksilver, but to some extent of silver.

The silver obtained is quite pure: it contains at San Dimas .994 of silver, .0033 of gold, leaving only .0026 for base metals. At Chihuahua it is .998 from the arrastra, and .990 from the treatment of the tails.* The bars weigh 35 kilos. The slags from the refining furnace, with the tails from the tanks for washing amalgam, and other products, are occasionally smelted in a shaft furnace with the addition of galena, and the lead is used in refining retort silver.

The loss of mercury in retorting varies from two to six kilogrammes per ton. The total loss of mercury in Mexico has been for many years calculated on the supposition that it requires a loss

* Mining Commissioners' Report for 1872, p. 438, Washington, D.C., 1873.

of a unit of mercury for every unit of silver obtained. This being a fixed amount, is called *consumido.* Any amount above this which is not recovered is called *perdida,* or loss, and is always attributed either to carelessness on the part of the workmen, or to mechanical losses during the operation. The losses, both the fixed and the variable, are always referred to the Mexican *mark,* which is equal to 248.83 grammes.

Consumido	248.83	grammes.
Perdida	124.42	,,
Total loss	373.25	,,

The loss of mercury for sulphuretted ores containing from $60 to $100 to the ton will be, under the most careful management, not less than four to five kilogrammes; in some exceptional cases it has been three kilogrammes for every kilogramme of silver extracted. The richer the ore the greater the loss. It may be averaged at one and a half kilogrammes for every kilogramme of silver produced. With ores containing large amounts of native silver the loss is proportionately much less, and sometimes even less in amount. The loss in silver varies from 20 to 25 per cent. of the assay value of the ore. Some amalgamators claim that they can save as much as 80 or even 85 per cent., but this is doubtful, even with the ores most easily treated. When the ores contain much blende and galena the loss easily reaches 25 to 30 per cent., and if in addition to this there is any amount of antimonial or arsenical sulphides, it will reach as high as 40 per cent. A part of this loss is, of course, counted with the loss of the amalgam, which is carried off in fine particles. It could easily be reduced by better appliances for catching the mercury, and better washing and concentration, which would catch a larger part of the pulp not acted on. But there is a mechanical loss as well as a chemical one, which must in any case be large. Just as soon as it is possible to introduce all the modern methods of concentration, the conditions will be such that other processes will take its place. Although much has been done to improve it, no process with large losses in the precious metal and excess of labour can hope to stand before increased facilities for transportation. When gold is contained in an ore which is sulphide, not more than 40 per cent. is re-

covered; when it is free, 75 per cent. is often saved.* The cost of the process will of necessity vary in different localities under dissimilar circumstances, and with ores whose composition is not the same. The results vary from year to year. Phillips gives the mean cost per ton for reducing these ores as follows:†

COST OF TREATING ORES.

Coarse crushing in dry stamps and subsequent fine grinding in arrastra	$1.90	
Manipulation in patio	4.50	
General expenses of management	1.20	
Repairs	1.20	
		$8.80
Sulphate of copper	$3.20	
Salt (1.6 quintals per ton)	6.50	
Quicksilver (11 oz. per 8 oz. of silver)	6.50	
		$16.20
		$25.00

Mr. Rul‡ gives the cost in detail as follows, for a much more recent period.

COST OF GRINDING ONE TON OF ORE.

Mules	$0.115
4 workmen	0.148
1 mule-driver...	0.055
Repairs	0.044
Night shifts	0.208
	$0.570

COST PER TON OF WORKING TEN QUINTALS EACH IN THIRTY ARRASTRAS.

Mules	$1.871
1 foreman	0.142
1 helper	0.077
3 feeders	0.099
5 arrastra men	0.219
3 watchmen	0.132
3 men	0.099
Bottom-stones	0.116
Grinding-stones	0.357
	$3.112

* "Engineering and Mining Journal," vol. xxxiii., p. 104.

† Phillips's "Gold and Silver," 1867, p. 357.

‡ "Engineering and Mining Journal," vol. xxxiii, p. 105.

PATIO WORKING, PER *Repaso.*

Mules	$0.029
7 workmen	0.021
	$0.050

14 repasos at 5 cents	.70
Salt	1.55
Sulphate of copper	0.96
Labour	0.17
	$3.38

SETTLERS AND DISTILLING.

Mules	$0.082
Various expenses	0.417
Charcoal	0.066
	$0.565

GENERAL EXPENSES.

Salaries	0.713
Rent	0.274
Repairs and miscellaneous	0.384
	$1.371

TOTAL COST OF WORKING PER TON.

Cost of grinding one ton of ore	$0.570
,, per ton of working 10 quintals in 30 arrastras	3.112
,, ,, patio working	3.380
,, ,, settlers and distilling	0.565
,, ,, salaries, rent, repairs, &c.	1.371
	$8,998

This estimate takes no account of the mercury lost, estimating this at about $7.00. This would make a cost of $16.00, a much lower figure than that given by other authorities.

The cost at the *hacienda* Sanceda in 1883 was;*

Pulverising in Chilian mills	$1.03
Grinding in arrastras	2.06
5 to 6 per cent. of salt	1.13
Magistral	0.80
1.71 lb. of mercury	0.96
General expenses	2.14
	$8.12

17,726 tons were treated with the average value of 17.11 oz. of silver to the ton. Of this, 4.37 oz., or 25 per cent., was lost. The

* Trans. Am. Inst. Min. Eng., vol. xiii., p. 370.

loss in quicksilver was 1 lb. per 7.4 oz. of silver extracted; the total cost in Mexican money being $8.12. The loss in silver appears to be about the same for poor and rich ores, as shown by the Table.

Assay Value of the Ore per ton.	Percentage of Silver lost.
32.22	10.0
37.00	10.0
42.60	6.1
47.60	5.6
99.60	7.0

Mr. Chism gives as the cost of working a $60 ore in 10-ton *tortas* as follows:*

COST PER TON OF 2000 LB.

Breaking per ton†	$1.53
Grinding	1.40
Scraping arrastra to get out the gold amalgam	.13
Carriage of slimes from arrastra to *patio*	.60
Mules hired	1.73
Labour, including driving and tending mules, spading and washing *torta*	1.80
Salt at 6 Mexican dollars per *carga* of 98.3 litres	2.80
Sulphate of copper at 0.25 (Mex.) per lb.	1.33
Charcoal for retorting and assaying at $0.37½ (Mex.) per *arroba*	.33
Quicksilver at $0.62½ (Mex.) per lb.	4.68
Salaries, general expenses, including keeping and feeding of mules	6.66
Repairs	2.33
Concentration of sulphurets	2.26
Total	$27.58

The expense of working with *tortas* of this size is much greater than if the pile were more than twice as large. When the *tortas* were of 19 tons and all the machines were driven by water power, the expense was as follows:

COST PER TON OF 2000 LB. IN WORKING A *torta* OF 19 TONS.

Breaking, grinding, and use of tools	$6.66
Amalgamators' wages	1.66
Scraping arrastra to get out gold amalgam	.16
Carrying and washing scrapings	.11
Concentrating tailings of ,,	.07

* Trans. Am. Inst. Min. Eng., vol. xi., p. 76.

† Breaking a ton of large ore costs $2.66, but, as the smalls are also worked, the average cost is as stated.

	Brought forward ...	$8.66
Carrying slimes from arrastra to *patio*	...	.42
Mules and keeping ...	...	3.72
Labour, spading, and mule-driving	...	1.60
,, washing *torta*	...	.56
Charcoal for retorting silver	...	.47
Concentrating tailings of *torta*	...	2.06
Materials, salt, 600 lb. at 8 cents	...	2.53
,, sulphate of copper, 125 lb. at 25 cents	...	1.65
,, precipitated ,, 25 ,, 66 ,,	...	.87
,, quicksilver ,, 133 ,, 62½ ,,	...	4.37
		$26.91

There is not only a very great saving in doing the work by power, but in custom mills, to which these last expenses refer, there is a considerable profit included in the cost, which will not be less than from $2 to $2½ per ton. It is astonishing how such a process has been able to retain its hold nearly three hundred years. In every country where it has been introduced, it, like many another historical process, has yielded before the advance of rapid means of communication, as this has in the United States and undoubtedly will in Mexico. It costs but little to carry it out, and can be worked on a large scale as well as a small one, the latter having only this disadvantage, that it increases the loss. The process requires peculiar conditions of climate, which adapt it especially to hot countries. On account of the climatic conditions it has been generally abandoned in the West, where it was formerly extensively used. When used at all at the present day it is on a very small scale, generally for experiments. It always works better on a hot day than on a cold one, in summer than in winter. The cheapness of the plant more than compensates for the time, as money in Mexico, where it is now chiefly used, is scarce; while time is of no value. Working the tails by the Von Patera process has, in some places, added to the yield in silver and increased the profits. The loss in reagents is easily put up with, as it is the only means by which the precious metals can be obtained. The method is only applicable to such ores as contain the silver native, or as chloride, bromide or iodide, associated as they usually are with highly oxidized substances. The presence of much sulphide renders the losses large. The process becomes difficult with the arsenio and antimonio sulphides, and impossible when there is much galena, blende, tetra-

hedrite, or bournonite in the ores. Not the least of the disadvantages of the process is the facility with which other people than the owners, may make a clean-up, the only protection against this being the difficulty of selling unrefined silver, especially in small quantities. In very large works where much capital is invested, the item of time is a matter of consequence, but there seems now to be no other process possible in Mexico, until transporation shall become less difficult.

The Cazo Process.

There seems to be no doubt that the *patio* process was in use in South America up to about the commencement of this century. It was still used there, to a very limited extent, until the year 1830, at which time it seems to have been quite generally given up, probably on account of the very large quantity of *negros* or sulphurous ores which began to be found. It was replaced in part by the *cazo*, or caldron method, which is still in use there, and in some parts of Mexico, and partly by a new method in which the copper bottom was replaced by an iron one, and finally by still another process, which while it imitated the pan amalgamation method so far as the machinery was concerned, added the chemicals which were supposed to form in the *patio* already prepared. These processes have a number of variations, all of which are known as the *Tina* processes, and are used for the treatment of impure ores. They have acquired great importance in South America, and may have some future in some parts of Mexico and of the United States We shall briefly describe all these processes.

The *cazo* process was invented in Chili, in the year 1609,* by a priest, Albaro Alonzo Barba, who, in his description of his own process, insists that the vessel in which the work is done should be made entirely of copper, though this was long since found not to be necessary. The ores to which this method is applied are the rich surface ores—chlorides, bromides, and iodides, which, if they are not rich enough, must be concentrated on the *planilla*. The process yields nearly the whole of the silver which is in them. The loss in mercury is from twice to two and a half times the total quantity of silver contained. The operation lasts not

* Percy's "Metallurgy of Gold and Silver," Part I., p. 656.

much over two hours, and gives tails which do not contain more than \$3 to \$4 to the ton, but it is generally only applicable to ores which contain \$80 and upward per ton, free from sulphur.

This process was formerly used in connection with the *patio.* The ores were first stamped and ground in the arrastra. This is done as a preliminary to a concentration. The grinding is not done so fine that there is danger of any large part of the silver being carried off in the washings. From the arrastra the pulp is carried to the *planilla,* where it is concentrated to such an extent that the concentrates do not represent more than two or three per cent. of the original ore. These concentrates are treated in the *cazo,* while the tails, if rich enough, were formerly treated on the *patio.* There are two processes known under the name of the *cazo,* distinguished from each other by the size of the vessel and the mechanical means of doing the work. The *cazo* is the smallest vessel. The larger one, constructed on exactly the same principle, is called a *fondon.* The process itself is very simple and rapid. It consists in boiling the concentrates, keeping them constantly agitated with salt and sulphate of copper, to which mercury is added, and then treating the amalgam.

The *cazo,* as originally invented, was a round vessel made entirely of copper, but was afterwards replaced by a vessel, at first made of stone, and then of wood, with a copper bottom turned up at the sides. This vessel was originally quite small. Its dimensions were: diameter above, 1 m.; diameter below, 0.60 m.; depth 0.45 m. The thickness of the copper bottom was 0.05 m. to 0.06 m. This was set over a fireplace without grate bars, or chimney, the smoke going out where the fuel was put in. A *cazo* of such very small dimensions could treat only about 50 kilos. at a time.

To treat the ore, water sufficient to make a thin pulp with the charge, was introduced. The fire was lighted, the water brought to a boil, and salt amounting to from 5 to 15 per cent. of the weight of the ore was then added. The workman then rubbed the bottom of the *cazo* with a piece of wood attached to a long pole, to keep the copper surface perfectly free. If the salt had been added before the ebullition of the pulp, it would have collected on the bottom, from which it would have been difficult to separate it. As soon as all the salt is dissolved, the first

addition of mercury is made. This will generally be introduced in several portions; one quarter only being added at first. In ten or fiteen minutes an assay is taken, with an open horn attached to a long handle, so as to pick out the heavy parts of the ore and amalgam. This is washed, and if the amalgam shows itself as a clear grey sand, *polvo*, the charge is ready for the second addition of mercury. The same quantity as before is added, the heat and movement being kept up. In an hour or two after the start, another assay is taken, and another addition of mercury made, and so on until an amalgam containing two parts of mercury for one of silver results. The operation is then considered as finished. The amalgamator, *cazeador*, takes a last assay, *prueva en crudo*, which he washes to get out the gangues, then adds a large excess of mercury to dissolve out the amalgam, separates it from the tails, and then examines it by rubbing it against the sides of the vessel to see if any of the ore remains. If it does, the operation must continue; if not, and the amalgam remains fluid, it is stopped. At the end of six hours the operation is complete. The muddy material is run off into outside receptacles, and what remains in the *cazo* is dipped out, and treated in *bateas*, with an amount of mercury equal to that which has already been used.

It is of the greatest importance, during the whole of the operation, to prevent anything from adhering to the bottom. If the salt was introduced before ebullition took place, it would collect on the bottom, and the apparatus would have to be emptied before it could be removed. It is more especially important to prevent any adherence of mercury, which would prevent the action of the salts of silver on the copper, and thus make the amalgamation progress very slowly. It would besides cause a great loss in mercury, as it alone, and not the copper, would reduce the silver salts. If the proportion of the mercury and silver are as two to one, no adherence of the mercury takes place.

The *cazo* was replaced by a much larger vessel,* 2.15 m. diameter above, and 1.80 m. below, and 0.85 m. deep, called a *fondon*. The bottom is made of impure cast copper, and is 0.18 m. to 0.20 m. thick, 1.80 m. in diameter, and 0.18 m. deep. On the

* "Ann. des Mines," Series 6, vol. xx., p. 216, Pl. III., Fig. 7.

inside of the rim of the basin a place is cut out to receive the staves, which rest on the bottom of the cut made in the rim of the copper basin. These staves are 0.70 m. long, and are held in position by iron hoops. All the joints between the copper and wood are made tight with clay, and then adobes are built up around the outside to a thickness of 0.45 m. In the centre a raised space is provided for the pivot of the upright arbor which carries two arms, one 0.45 m. from the bottom, of a little less diameter than the interior of the *fondon,* and the other at 0.85 m., projecting beyond it for the purpose of hitching a single mule to it. The lower arm carries two pieces of copper, each of which weighs 140 kilos., which are used as mullers. They must be so arranged as to rub over the whole surface of the copper bottom to keep anything from becoming attached to it, as it would otherwise be impossible to grind with such soft materials as a copper muller. The whole is placed over a furnace with grate bars, on which the inferior fuel of the country is used. Such a *fondon* will last for ten years. The cost is*

60 quintals of copper for bottom, at $20	$1200
Two mullers...	120
25 staves at 3 reels	9
Furnace	40
Woodwork for the mules	10
House	200
	$1579

When everything is ready, the *fondon* is charged with 500 to 600 kilos. of rich ore, and 30 to 40 kilos. of the powder of unwashed ore, and sufficient water to form a thin mud. Fire is kindled on the grate, and the muller set in motion. At the end of two hours the material is boiling; 52 kilos. of salt, or about 10 per cent. of the weight of the ore, are added. This relatively large amount is necessary, as the success of the process depends to a great degree upon the quantity of salt used, and the velocity of the mullers. With the richest ores, the quantity of salt does not exceed 25 per cent., and whatever may be the necessity for it, the number of turns of the muller will hardly exceed ten. About half the weight of silver contained in the ore is then added in mercury, and the mullers set in motion at the rate of ten turns

* "Ann. des Mines," Series 6, vol. xx., p. 217.

per minute. The amalgamation commences at once. At the end of an hour an assay is taken from the bottom, taking care to take it ahead of the muller. If the amalgam washed out looks like light grey sand, it is composed of two of mercury for one of silver; the same quantity of mercury is again added, and at the end of an hour another assay is made, and so on, until the amalgam, even after it has been worked in the *fondon* for half an hour, shows an excess of mercury. The *prueva en crudo* is then made, and if any ore is found, the operation is continued half an hour without any addition of mercury. At the end of six hours the operation will generally be finished.

If there is an excess of mercury, there is danger that the sides of the vessel will be attacked; if there is no excess, but if the velocity of the muller is decreased, the copper and mercury become alloyed, and the bottom of the *fondon* becomes coated with a very thin coating of silver amalgam which is very difficult to remove. As the copper surface is much diminished, the operation is very considerably lengthened. There is also danger that the mercury will flour, and the loss in silver will be very great. There is only one remedy for this, which is to empty the *fondon,* and scrape the bottom clean. It is very easy to prevent this accident by adding the mercury carefully and in small quantities at a time, and by keeping up a uniform but rapid motion of the mullers. With these precautions, the work is very nearly independent of the skill and intelligence of the men. The results are quite uniform, and are obtained in a very short time.

As the reactions are not performed at the expense of the mercury, there is no occasion for any loss of it. If the operation is well carried out, all the mercury used should be collected at the end of the process; but this is never done. Some of it is floured, some of it volatilized, so that the loss is counted at about two per cent. The reason why there is such a small loss probably is, that the work is done hot. The loss in silver is variable. The ores almost always contain sulphides more or less rich in silver, which are not acted on by this method. The tails vary from \$25 to \$40 to the ton, so that the *fondon* process can generally be used only as a preliminary method, and the *patio,* or some other process, is usually associated with it. The residues

remaining in the *fondon* consist for the most part of the oxides of lead and iron, and some sulphurets containing silver and floured mercury. These are washed in large wooden bowls in a water-tight vat, adding as much mercury by weight as there is material to be treated, in order to collect the flour. The amalgam is treated as usual.

In Mexico, the slimes which have been removed are put into catch-pits where the excess of water evaporates. They are then made into small *tortas*, which are trodden by men. Two to two and a half per cent. of salt is added to them, but no magistral, for the water coming out of the *fondon* contains enough copper salts to do the whole of the work. The amalgamation is conducted as usual, except that it is very slow, lasting often as long as three months. The loss in silver is as much as 20 to 25 per cent. The mercury used is 125 to 150 per cent. of the silver contained. This method is one of the most rapid and least expensive of the *cazo* processes. The cost is given below :*

Cazeador (amalgamator)	$0.500
Atizador (furnace man)	0.280
Wood for heating the furnace	1.562
Salt, 75 lb., at $6 for 300 lb.	1.500
Mules	0.187
Mercury, two per cent., loss	0.416
Cost of distillation, &c.	0.250
	$4.695

In a single operation 1200 lb. of ore are treated, which is 9.33 reals per charge.

If to these the expenses of dressing and concentration on the *planilla* are added, calculating the expenses in grammes of fine silver per ton, we have as follows :

			Grammes.
Crushing with mule power			17.360
,, in arrastra			57.860
Washing on the *planilla*			17.360
Amalgamation	Labour	34.720	
	Power	8.657	
	Fuel	72.313	
	Salt	69.443	
	Mercury ...	19.258	
	Distillation ...	11.573	
			215.964
Cost in grammes per ton			308.544

* "Ann. des Mines," Series 6, vol. xx., p. 221.

The very friable nature of the gangues has much to do with the small cost of the concentration. The cost elsewhere in grammes is :

	Grammes.
Cost of extraction and sorting	92.59
Transportation	69.44
Treatment	231.47
	393.50

This includes the cost of mercury, and shows a minimum for the metallurgical treatment. The treatment of these ores gives 400 grammes in the *cazo*, which pays the cost, the profit being in the treatment of the tails.

An attempt was made in Chili* to treat rich sulphurous ores with sulphate of copper and salt, but though it was a rapid process, and the tails were poor, the enormous losses in mercury caused it to be entirely abandoned. It was replaced by a method no longer used, but which is interesting as showing how another grew out of it.

The ores upon which the process was used are the rich bromides, chlorides, and iodides of the upper part of the veins. The gangue was oxide of iron, the carbonates of baryta and lime, and some clay. They contained generally from $300 to $400 of silver to the ton. When such ores as these became rare, some other process had to be used. This method caused the almost complete abandonment of the *cazo* process proper; and it was not until the ores became so very poor that it was no longer applicable, that it was replaced by the process now used in the vicinity of Copiapo.

The ores were reduced to pulp by methods analogous to those used in the *patio* process, from which this one originated. The pulp is carried away by a stream of water to settling-tanks 2 m. in diameter and 3 m. deep, made of sheet iron, the number in use depending on the size of the works. As fast as one of these settling-tanks is full, the stream is turned into another, and so on. The tanks, when full, are left from eight to twelve hours. The clear water above is then run off, and the mud below carried to the *tinas*. These are wooden tanks with cast-iron bottoms. They are 1.80 m. in diameter, and 1.20 m. deep. In the centre is an axis which carries a muller, which runs on or close to the

* "Revue Universelle des Mines," Series 1, vol. xxxi., p. 489.

bottom of the *tina*. This machine was undoubtedly suggested by the arrastra. The charge for each *tina* is one and a half tons of pulp. It is introduced into the *tina* while the muller is still. Mercury is added, to about twenty times the amount of silver contained in the ore, and the muller put into very slow motion, not over four times a minute. At the end of twenty hours the amalgamation was supposed to be completed. A stream of water was then introduced, and the light particles were thus carried off. When the water ran clear, the particles being too heavy to remain suspended in it, the mercury and amalgam were removed through a hole made in the *tina* for that purpose, and collected in a cast-iron vessel called a *cocha*. A complete operation, including the grinding, lasts about 60 hours. The cost for ores yielding $80 to the ton is $10 per ton, including the loss in mercury. The tails usually contain from $8 to $10 a ton. They are not allowed to contain more than $25 to $30. As the ores themselves are very pure, the silver obtained is about .990 fine. So long as the ores were rich and pure, little was done to improve the process, but as they became poorer and more impure, the tails grew constantly richer, and it became necessary not only to treat them, but to treat the poor ores, *desmontes*, which had been thrown aside as not worth treatment. Barrel amalgamation was tried, but failed, as did also the attempt to chlorurise the ores and dissolve out the chloride of silver, as the ammonia cost too much Recourse was then had to the abandoned *cazo* process, which, with a number of modifications, proved successful.

Another and very simple process* which grew out of this preceding one is applicable to all the ores of silver except argentiferous sulphides of copper, galena, or blende, and to ores which contain more than one per cent. of free arsenic, which causes great losses in the mercury. The inventor of it is not known, but it has been in constant use about Copiapo since 1862.

The ores must be carefully sorted, so as to separate them into different classes, keeping the especially rich ores by themselves, as these are worked much more rapidly than those of lower grade. The difference of time in the treatment of the different ores more than makes up for the trouble it costs.

* "Revue Universelle des Mines," Series 1, vol. xxxi., p. 493.

The rich ores, including the sulphides, are treated in copper tanks with sulphate of copper, salt, and mercury. The solutions are all made by steam, and beforehand, five per cent. of the weight of the mineral being added in salt. The sulphate of copper solution is made up to 20° B., and to it salt is added until no more will dissolve. The sulphate of copper is in this way transformed into chloride of copper, and the soda to sulphate of soda. When the liquor is saturated, it is decanted into large wooden tanks, and metallic copper, usually old copper sheathing, is put into the liquor, which is heated to ebullition by a current of steam at a pressure of three atmospheres. This causes the copper to be attacked, and a sub-chloride of copper is formed which is used in the process. The operation is finished when, by taking about 50 c. c. of the liquor and putting it into a litre of water, the oxychloride precipitates as a white powder, leaving the liquor colourless. The sub-chloride is then formed. The salt requires one vat, the sulphate of copper two, and the sub-chloride one, in their preparation. When the sub-chloride is formed it must be used as soon as possible, to prevent the formation of the oxychloride, and in order to do this as far as can be done, the solution is slightly acidulated with sulphuric acid.

A cast-iron Chilian mill, *trapiche,* each wheel of which weighs four tons, is used for grinding the ores. The bottom of the mill is called *solera.* This is usually made of cast iron, but sometimes of steel. The mill turns at the rate of ten to twelve turns a minute. The ore, which is ground sufficiently fine, is carried off by a current of water, the quantity of which is regulated according to the fineness to which the ore is to be ground. This water is made to pass through slime-pits five metres by two metres, and one metre deep, and must run off perfectly clear from the last one. When one of the tanks is full, the stream is turned on to another. The full one is left for eight to ten hours. The clear water is then drawn off, and the pulp thrown out with shovels upon an area called *cancha,* to dry. When the ore is sufficiently dry, it is charged into barrels similar to the Freiberg amalgamation barrels. They are of different sizes, their capacity being from one to four tons, the larger the better. Those which hold four tons are 1.80 m. by 1.50 m., the staves are 0.075 m. thick. To

the four tons of ore, enough of the salt solution is added to form a thick mud. The quantity of magistral to be added depends on the kind of gangue, much more being required for carbonate of lime than for clay or oxide of iron, as the former decomposes the sub-chloride. For an ore of about $80 to the ton, and a variable gangue, 28 to 30 litres of the magistral are added. The barrels are turned from twenty minutes to half an hour to make the mud quite uniform. Mercury amounting to from twenty to twenty-five times the quantity of silver contained, is then added. If there is a large amount of chloride or bromide of silver in the ore, 25 per cent. of the weight of the silver contents of the ore is added in lead. This is amalgamated with mercury before it is introduced, and has for its object to prevent the formation of chloride and bromide of mercury, and a consequent loss. Lead is very easily attacked by the chloride and bromide set free—much more easily than mercury. This saves the mercury from being lost as chloride, and also prevents a mechanical loss, as the chloride of mercury, once formed, envelopes the globules of mercury and prevents both their coming together in a mass and their action on the silver. Besides this, the mercury is much more easily reduced to a powder by this means, and is kept so, causing a great loss. This simple device of using lead reduced the loss in mercury, when the chloride and bromide ores were used, from 150 per cent. to 25 per cent. As soon as the mercury is introduced, the barrels are turned at the rate of four to five turns a minute for six hours. The operation is then complete. Water is added in considerable quantities, the barrel being turned for a short time, and the tails, amalgam, and mercury discharged as in the Freiberg process. The amalgam recovered is not pure. It contains oxide of copper, produced by the action of the lime of the gangue on the chloride of copper, and the sulphides of copper produced by the action of the sulphate of copper on the sulphide of silver. These must be separated, the one by mechanical means, the other by chemical action. The first is done in a *tina*. The amalgam is charged with ten per cent. of fresh mercury. Water is added, and the muller is made to revolve at the rate of sixteen turns a minute. When the water which comes off is entirely clear, all the sulphide,

and a part of the oxide of copper, will have been removed. To remove the oxide, all the water of the *tina* is run off, and 2 per cent. of carbonate of ammonia is added. The muller is revolved for five hours. At the end of that time it is stopped, and the amalgam washed with water. If this has been properly done no oxide will be left. The amalgam is distilled in a *capellina*. The mercury which is strained from the amalgam becomes little by little quite impure. After it has been used five or six times, it amalgamates very slowly. It is then purified by adding to it 20 grammes of sodium amalgam for every 100 kilos. of impure mercury.

The resulting silver, *pina*, is refined in a reverberatory furnace. It contains some arsenic, which is extracted by the iron of the tools, and floats on the surface of the bath and is removed. The method of refining does not differ in other respects from that used elsewhere. The silver obtained is 980 fine. By this process, tails which do not contain more than \$6 to \$8 to the ton, and ores of from \$10 upwards, are worked. When the ores do not contain more than \$80 to the ton, the tails do not contain much more than \$2 to \$3. Plenty of good water is a necessity for such works, both for purposes of washing and for power if possible.

To treat eight tons of ore in twenty-four hours, requires an area of 500 square metres for the ores, and one of 1000 square metres for drying the pulp; two Chilian mills requiring about six horse-power; two settling tanks, and two amalgamation barrels requiring about eight horse-power; a vat to collect the water from washing the barrels, to recover the floured mercury; one trough for washing the amalgam; one distilling furnace; one reverberatory furnace for refining the silver; a tank for the preparation of the magistral, with a three horse-power boiler attached; two vats for dissolving the sulphate of copper; a vat built with hydraulic cement to make the salt solution, with a boiler for boiling it; a syphon for clarifying the liquors, which must all be treated with lime to precipitate the copper contained in them:—these constitute the machinery and apparatus for the works. The cost of treating a ton of ore of about \$40, not including interest nor sinking fund, would be—

Crushing	$1.60
Mercury, magistral, and salt	4.00
Purifying the amalgam	.04
Distillation	.04
Fusion and firing	.09
Various expenses	1.00 to 1.10
	$6.87

The whole operation is very simple—quicker, and with less loss, than the barrel, more certain in its reactions than the *patio*, and applicable to almost all the ores found in Chili. It is even cheaper, under some circumstances, than the lead fusion.

Still another variety of the *tina* process has been invented, by Francke, and is used at the Huanchaca and Guadaloupe mines at Potosi, Bolivia, where it has been very successfully worked for some years. It is of great interest, as it includes many of the best points of the Cazo and Pan processes, and as it may be used for ores of a great variety of yield, it may possibly have some application in the United States. It is used in South America on ores containing 150 oz. and over to the ton, but may be used on much poorer ores. The ores in Bolivia are highly sulphuretted and are very impure. They are first carefully hand picked, and sorted into three grades, as at Chihuahua (p. 267.) Those containing from 1000 oz. to 2000 oz. are generally sent to Europe to be treated. The second-class ores yield from 250 oz. to 500 oz., and the third from 100 oz. to 250 oz. The second and third-class ores are either kept entirely separate, or are mixed together, in such quantities as to make an average yield of silver, and of such quantity and quality of gangue, as may be found best to treat. As by the process of hand picking, the ores are already small, they do not need to pass through a crusher, but are taken directly to the stamp mill, where they are crushed dry, and made to pass a 40-to-the-inch sieve. As the ores of Potosi contain much sulphur and considerable quantities of lead, they are roasted in double hearth reverberatory furnaces, one hearth being above the other. The furnaces are quite small, costing only about $500 each, and are capable of finishing only 2 to 2½ tons in twenty-four hours. The ore is first roasted on the upper hearth to drive off the sulphur and the volatile materials, as it contains arsenic, antimony, zinc and lead, besides variable propor-

tions of copper, sufficient, in most cases, to form a large part of the copper compounds, which are necessary for the reactions required in the subsequent treatment. When the volatile metals are for the most part eliminated, the ore still retaining considerable sulphur, is raked on to the lower hearth, and about 400 lb. of salt is added. This salt costs $0.55 per lb. This is not sufficient to transform all the metals into chloride, but this is not necessary as a considerable part of the chloruration is subsequently effected in the *tina*. The cost of working is quite heavy as the fuel is very expensive. Three kinds of it are used in order to get the necessary temperature at the least cost. This cost is made up as follows:

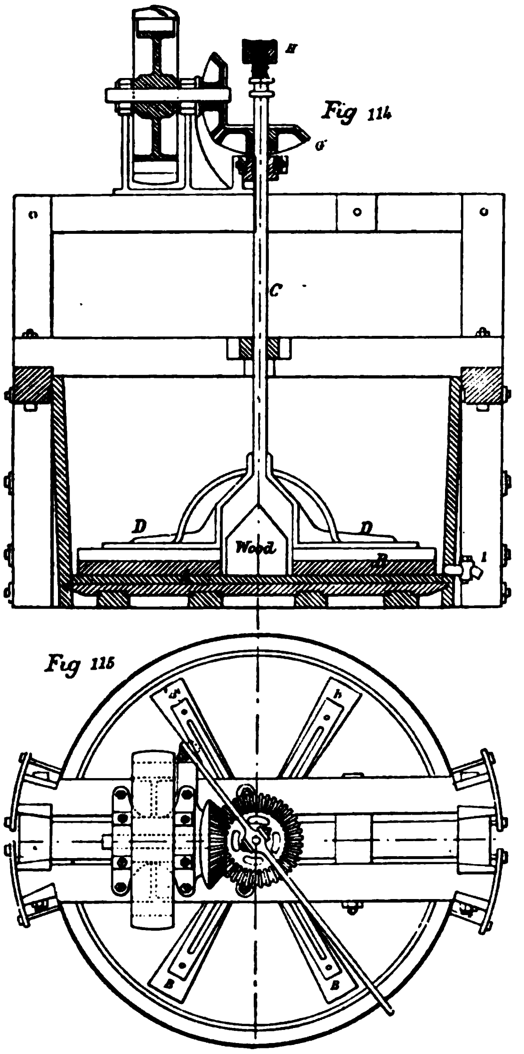
Tola, a coarse shrub, 134 kilos. at $0.01	$1.34
Yareta, a resinous moss, 200 kilos. at $0.025	2.50
Torba, turf, 500 kilos. at $0.0065	3.25
Labour	.75
Total cost	$7.84

One man is all that is required to do the work of two furnaces; he is paid $0.75 for twelve hours' work. The temperature is not high, and the repairs to the furnaces are very slight. The ore is drawn on to the floor to cool and is then taken in cars to the ore hoppers from which it is discharged into the *tina*. Each *tina* has its own hopper. The charge from the roasting furnace fills it, and a slide valve below allows of discharging it from the hopper directly into the *tina*.

There are two varieties of the *tina*,* which are shown in Figs. 114 to 117. They resemble the pan, but are constructed differently, though the idea was undoubtedly taken from the pan. The machine combines the advantages of the *cazo*, and the pan is much more easily constructed and managed than either. The two *tina* do not differ from each other in any essential principle, but only in slight details of construction. Figs. 114 and 115 is the one used at Huanchaca, and Figs. 116 and 117 the one used at Guadaloupe. The *tina* is a strong wooden tub bound with heavy iron hoops. It varies in diameter from 1.80 m. to 3 m. It is usually 1.60 m. in depth, the size depending on the quantity of ore it is desirable to treat at a time. The bottom is

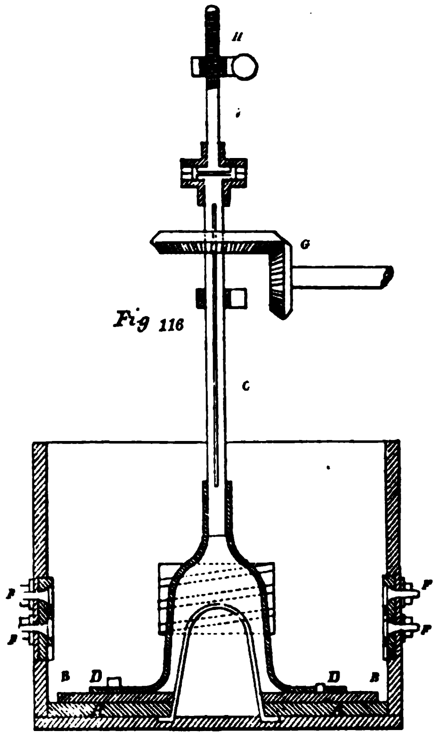
* Engineering, vol. xxxviii., p. 173.

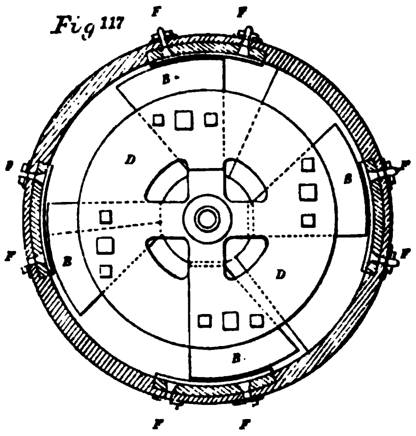
either supported on four heavy cleats, as in Fig. 114, or is made entirely flat. They are placed on platforms in a row, so as to be controlled by the same line of shafting. The bottom is covered on the inside with plates of copper, the entire diameter



of the *tina* and from 0.06 m. to 0.08 m. thick. In the centre is an upright shaft C, which is carried by gear wheels G, and supports in the tub a copper muller B; this is securely bolted to the stirrer D. The method of supporting the muller is shown in

the drawings. At Huanchaca, Figs. 114 and 115, the muller is attached to four arms, which have but a small surface, but at





Guadaloupe, Figs. 116 and 117, it is fastened to a single bell-shaped piece, very much like the shape of the cone of a pan muller, to

the underside of which the copper plates B are fastened. As these cover more than half the surface of the bottom of the *tina,* it is undoubtedly the best disposition. The muller can be raised or lowered when necessary by the screw H at the top of the shaft. On the inside, above the muller, Figs. 116 and 117, are four copper plates or wings, which are slightly inclined, and which are securely fastened on the outside by the bolts F. In order to give more effective surface, these plates are ribbed. On the bottom of the *tina* there is a large wooden stop-cock I, Fig. 114, to draw off the amalgam and to empty it when the process is complete.

To carry out the process, the *tina* is filled about 0.30 m. deep with water, which is introduced from a main, running over the top of it, with a flexible hose for each pan. From 130 kilos. to 160 kilos. of salt to the ton of ore is then introduced, and the muller run at the rate of forty-five revolutions per minute. Steam is introduced into the liquid, which is kept agitated for half an hour, or until the contents of the *tina* are boiling, the whole charge of ore is then introduced from the hopper. At the end of another half-hour about 50 kilos. of mercury is introduced. After some time tests are made, and further quantities of mercury added as it is wanted. The amounts of mercury required vary with the richness of the ore. It is generally added in three equal additions, the first is put in at the commencement, the second at short intervals about the middle, the last at the end of the process. For ore of about 200 oz., only 50 kilos., for ore of 150 oz. to 175 oz., about 30 kilos., and for low grade ores of 20 oz. to 30 oz., about 20 kilos., is used in each of the three additions. The charge of ore remains about the same, or 2½ tons. The reactions which take place are about the same as those in the pan, the copper being furnished partly from the ore and partly by the abrasion of the muller and dies. As the reactions are all done hot, they are certain to be most efficient. The whole operation lasts from eight to twelve hours. When the assay shows that it is ended, the stop-cock I, Fig. 114, is opened, and the amalgam collected. The tails are examined and treated in the usual way. The power required for each *tina* is from 2½ to 3 horse-power, or about a horse-power per ton of ore to be treated.

The amalgam is carefully washed with water to remove any sulphides or particles of ore which may be in it, and then treated

Fig. 118

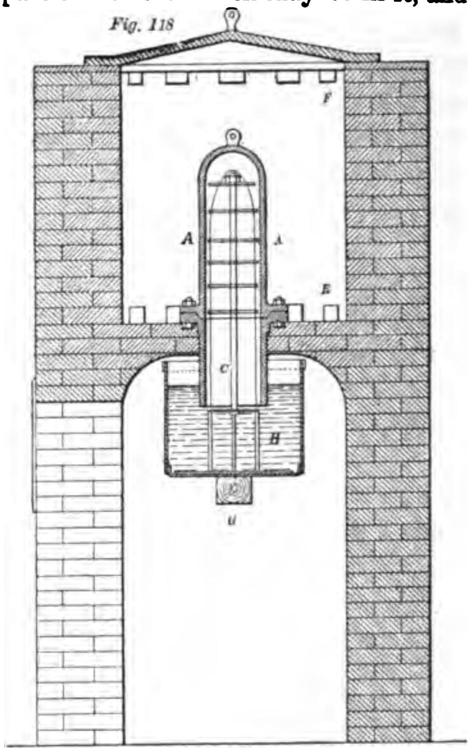
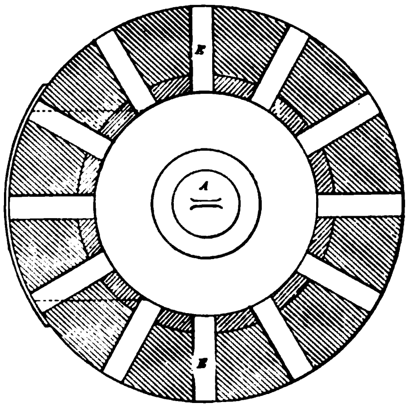


Fig 119



with an excess of mercury, as in the other processes. It is then treated in an hydraulic press to remove the excess of mercury.

The cakes of metal are forced, in the press, into very nearly the shape of a pine cone, and the cake of pressed amalgam is called *pina.* When all the amalgam has been pressed, the *pina* are weighed, after which they are distilled in a *capellina* of modern construction, shown in Figs. 118 and 119, several of which are set in a row. In front of them is a small railroad for carrying the amalgam to the furnaces and the retort silver to the weighing house. The furnaces are circular, and are lined with firebrick; they are 1.20 m. inside diameter, and 1.30 m. high. Below each one of them there is an arch chamber of the same diameter, and about 1.75 m. high. In the centre of the arch supporting the two chambers is an opening 0.30 m. in diameter, into which a cast-iron pipe, open at the bottom, is set, to which the *capellina* proper, A, is permanently bolted. The furnace itself is closed with a cast-iron lid, which has a ring to remove it easily. The *capellina* has also a ring to remove it when it needs to be replaced. It cracks after use, and must be detached from the lower part when it is to be removed; the lower part is not affected by the heat, and consequently lasts a very long time. Round the bottom of the furnace there are a series of holes E, for the admission of air and for stirring the fire, and on the top holes F, for the escape of the products of combustion. In the vault beneath the furnace is an iron tank B, two-thirds full of water, which is supported on a beam G, into which the lower part of the *capellina* dips below the surface of the water. Inside the *capellina* is a stand of iron with a series of horizontal shelves supported on a frame on the bottom of the tank B. These shelves do not quite touch the sides of the *capellina.* When the *capellina* is to be filled, the beam G is lowered, and the tank removed from the vault, and with it the interior shelves. These are packed with the *pina.* The tank with the *pina* so packed is then brought into the vault, raised to its place, and held there by the beam G. A fire is then made in the furnace, llama dung being used. It is an excellent fuel for this purpose, as it gives a smouldering but a constant fire, and sufficient heat for distilling the amalgam. It costs $0.63 for 75 kilos. The loss in mercury is about 4 oz. to every pound of silver. The volatilized

mercury is condensed and is collected in the tank B. When the operation is complete, the furnace is allowed to get cold, the tank is lowered, the *pinas* removed, and the silver melted and run into bars. The process is simple and applicable to very impure ores. In most of its essential details it resembles pan amalgamation, but ores can be treated by it that could not be treated in a pan. The process of distilling the amalgam is very primitive. The *capellina* furnace, however, is a very great improvement on the one formerly used in South America.

GLOSSARY OF TERMS USED IN THIS CHAPTER.

Adobes,	Sun-dried bricks.
Arrastra,	Mexican mill for grinding ore.
Arrastra de cuchara,	A spoon *arrastra.*
Arrastra de marca,	A large *arrastra*
Arrastra de mula,	An *arrastra* worked by mules.
Arroba,	Mexican weight of 40 lb.
Atizador,	Furnace-man.
Azogue,	Quicksilver.
Azogueria,	The mercury-house.
Azoguero,	The amalgamator.
Baño,	Excess of mercury used in the *torta.*
Batea,	A bowl.
Batea apuradora,	Wooden bowl floating on the *pila apuradora* to receive the *cabezilla.*
Bolichar,	Treatment in *boliche.*
Boliche,	Bowl for concentrating.
Bollos,	Triangular bricks of amalgam.
Bonanza,	Rich pocket in a vein.
Cabezilla.	Residue after washing the *torta.*
Cabezuela,	Concentrates rich in gold and silver.
Cajetes,	Walled receivers for the ground slimes (See *Lameros*)
Caliche,	Feldspar.
Calichoso,	Feldspathic.
Calor de frio,	Steam caused by the difference between the heat of the pile and of the air.
Cancha,	Space for drying slimes.
Capellina,	Bell covering *bollos* while distilling off the mercury.

Carga,	Mexican weight of 300 lb.
Cazeador,	Amalgamator.
Cazo,	A vessel with a copper bottom, for heating and amalgamating the ore.
Ceja,	Silvery-white amalgam.
Chuza,	Washer or settler.
Colas,	Brown sulphurets above the *polvillo* in the *boliche*.
Cocha,	A cast-iron vessel.
Colorados,	Coloured ores containing silver.
Comalillos,	Calcination furnaces for making magistral.
Consumido,	Fixed loss of mercury.
Contratanque,	Second settling tank.
Copela,	Dry amalgam in bag after draining.
Copelilla,	Zinc blende.
Cuchara,	A hollowed spoon-shaped float on the *arrastra*.
Debil,	Term applied to amalgam when very fluid.
Descargadora,	Discharging tank, from which the slimes are run off last.
Desecho,	Broken-up mercury. The attacking of the amalgam by the sulphur, &c., causing loss of silver.
Despoblado,	Ore with much gangue.
Desmontes,	Poor ores.
Ensalmorar,	The addition of salt.
Estrujon,	Amalgam strained from the mercury collected in the basin of the furnace.
Estufa,	Stove for evaporating the mercury from the amalgam.
Ferro blanco,	Arsenopyrite.
Fondon,	A large *cazo*.
Fuerte,	Strong; applied to amalgam needing more mercury.
Galeme,	Lead cupellation furnace for silver.
Galera,	A long shed on each side of the *patio*.
Granza,	Coarse sand from stamping-mill.
Granza de llunque,	Third class ore.
Guija,	Quartz.
Guijoso,	Quartzose.
Hacienda,	Establishment for treating ores.
Ijadas,	Assays of two to five pounds.
Incorporo,	Mixing the magistral and mercury in the *torta*.
Insalmoro,	Salting the *torta*.
Jicara,	A small vessel or bowl in which the assay sample is washed and the amalgam tested.
Jales,	Tailings.
Lagune,	A small lake.
Lama,	Slimes.
Lameros,	Slime pits; walled receivers for the ground slimes. (See *Cajetes*).
Lava,	Washing the *torta*.
Lavadero,	The ordinary settler; washing apparatus.
Limadura de plata,	Dry silver amalgam.

Lista,	Tail of impure mercury.
Magistral,	Roasted copper pyrites, sulphate of copper, &c., used to reduce silver ores in the *torta*.
Manga,	Canvas bag to drain amalgam.
Marc,	Mexican weight for weighing silver and gold, 8 oz.
Marmajas,	Concentrated sulphides,
Metal calichoso,	Feldspathic ore.
Metal de beneficio,	Second class ore worked on the *patio*.
Metal de exportacion,	First-class ore ready for sale.
Metal hecho,	Hand-picked rich ore.
Metal de primera clase,	First-class ore ready for sale.
Metal gabarro,	First and second-class ore, from the size of an egg to that of an orange.
Metal granza,	Fine ore, smalls.
Metal de labores,	Smalls from the workings of the mine.
Metal de llunque,	Smalls from the cleaners.
Monton,	Mexican weight varying from .75 to 1.62 tons.
Molino,	Stamp mill for ore.
Morteros,	Stamping mills.
Negros,	Black ores. Generally sulphides of silver.
Oroche,	Bullion after retorting.
Pasilla,	Dry silver amalgam.
Patio,	Amalgamation court.
Perdida,	Loss of quicksilver beside the *consumido*.
Pila,	A trough of hide.
Pila apuradora,	Tank to receive the residues from the washing-tanks.
Pina,	Pressed amalgam or retort silver.
Planilla,	Inclined platform to concentrate tailings.
Planillero,	Operator on *planilla*.
Plata,	Silver.
Plata cornea amarillia,	Iodyrite.
Plata cornea blanca,	Kerargyrite.
Plata cornea verde,	Embolite.
Plata mixta,	Alloy of gold and silver.
Plata negra,	Argentite.
Plata pasta,	The spongy bars of silver after retorting.
Plata piña,	Silver after retorting.
Plata verde,	Bromyrite.
Platillo,	Earthen plate for testing the slimes.
Plomo,	Galena.
Polvillo,	Rich black sulphurets left on *planilla*.
Polvo,	Fine grained amalgam from *cazo*.
Precipitado,	Metallic copper precipitated by iron or zinc.
Prueva en crudo,	An assay from the *cazo*.
Quebradero,	Breaker or crusher.
Quemadero,	Distillation furnace ; retort.
Quemazon,	Black decomposed ore.
Quintal,	A hundred pounds.

Raspa,	That portion of the precious metals obtained by scraping the *arrastra,* or the *patio.*
Raspadura,	Scrapings.
Raspando,	Scraping; removing the amalgam from the *arrastra* by scraping.
Relaves,	Material remaining after the washing of the *tortas.* (See *Polvillo.*)
Rendido,	Term applied to *torta,* when the amalgamation is concluded.
Repaso,	Treading of the ore in the *torta.*
Rosiclara,	Ruby silver.
Saltierra,	Impure salt from lagunes.
Solera,	Cast-iron bottom of a Chilian mill.
Tahona,	A spoon *arrastra.*
Tahonero,	Man in charge of the *tahona* or *arrastra.*
Tanque,	First settling-tank.
Tentadura,	Assay.
Tierras de labores,	Smalls from the workings of the mine. (See *Metal granza de labores.*)
Tierras de llunque,	Smalls from the cleaners. (See *Metal granza de llunque.*)
Tina,	A circular tank; a round dolly-tub.
Tina cargadora,	Tank into which the slimes are first discharged.
Torba,	Peat.
Tola,	A course shrub.
Torta,	Heap of slimes on the *patio.*
Tosa,	Grinding space in the *arrastra.*
Trapiche,	Chilian mill.
Trilla.	Heap of slimes on the *patio.* (See *Torta.*)
Voladora,	A muller.
Voltear la torta.	Spading: turning the *torta.*
Yareta,	A resinous moss.

CHAPTER VII.

BARREL AMALGAMATION.

THE PELICAN MILL, COLORADO.

THE Saxon method of amalgamation, or the Barrel or Freiberg process as it is called, has never been extensively used in the United States. It has been practised in a few mills for a short time, but has always given way before the more successful process of pan amalgamation. One of the few works where it was used in 1875 was the Pelican Mill. These works were situated in the Rocky Mountains, at an elevation of 8300 ft., in the centre of the town of Georgetown, Clear Creek County, Colorado, fifty miles from Denver. In 1875 the terminus of the Colorado Central Railway was fifteen miles from Georgetown. The mill treated ore from the Pelican Mine, and such ores as were offered in the market which were suitable. Most of the ores, however, came from the Pelican Mine, and were distinguished by the miners as "sulphides," which are heavy ores, and "sulphurets," which are mostly light or oxidized ores. At the mill they are called "Pelican straight" and "Pelican light." The first is a very heavy ore, containing at least 16 per cent. of zinc, 15 to 20 per cent. of lead, besides pyrites, quartz, and feldspar. It contains on an average from 150 oz. to 200 oz. of silver, and rarely went as high as 270 oz. The average was from 130 oz. to 160 oz. The copper and iron pyrites contained about ¼ oz. of gold, but the pyrites was in so small quantity that there is only a trace of gold in the ore, and hardly an appreciable amount in the bullion. The "light" ore is composed of oxide with very little sulphur. The plant of the mill consists of one Dodge crusher, five Brückner cylinders, two ball crushers, five amalgamation barrels, and two dolly tubs. The mill can treat 15 tons of ore a day.

The process consists of:

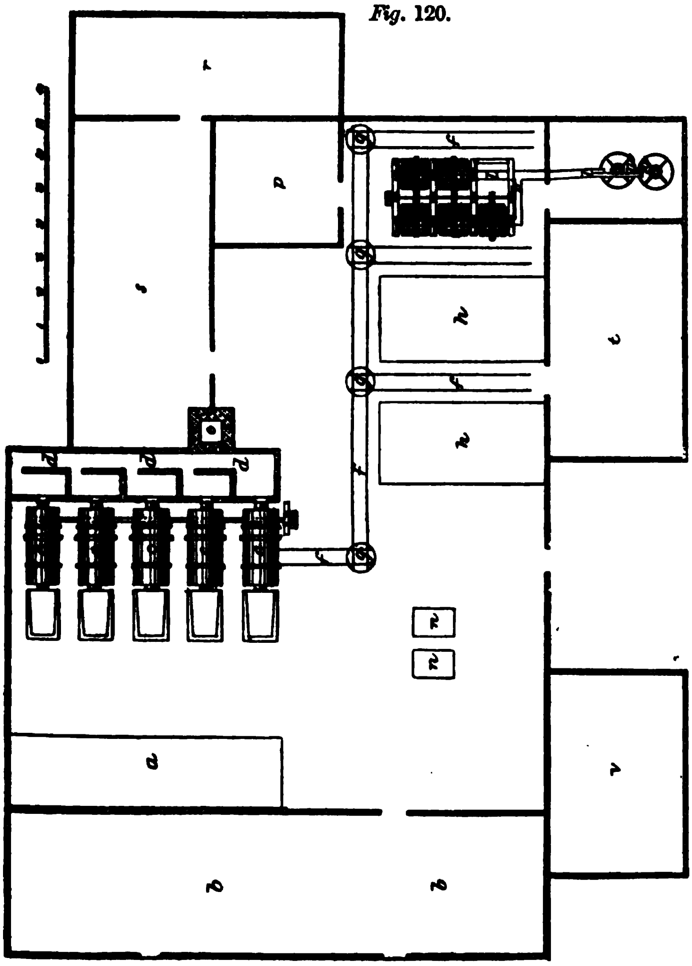
1. Crushing and grinding the ore.
2. Roasting the ore in the Brückner cylinders.
3. Amalgamating the roasted ore in barrels.
4. Distilling the amalgam, and fusing the silver.

The following Table gives the prices paid for ore in the year 1874:

	No. of oz. per ton.	Price paid for each oz. above 35.		No. of oz. per ton.	Price paid for each oz. above 35.
		$			$
Up to	299	1.05	Up to	600	1.03
,,	300	0.93	,,	700	1.05
,,	400	0.97	,,	800	1.07
,,	450	0.99	,,	900	1.08
,,	500	1.01	,,	1000	1.09
,,	550	1.02	,,	2000	1.10

No deduction is made for zinc, lead, or copper, and nothing is paid for ores of 35 oz. and below. The plan of the mill as it was reconstructed in 1874, while the old mill was being torn down, is shown in Fig. 120:

Fig. 120.



a. Space for drying furnace
b. Ore house
c. Brückner's cylinders
d. Flue
e. Chimney
f. Railroad
g. Turntables
h. Brick cooling floor
k. Wooden barrels
l. Discharge trough
m. Dolly tub
n. Ore crusher
p. Room for cleaning amalgam
r. Boiler room
s. Engine room
t. Water-wheel room
v. Tail house

I. Crushing and Grinding the Ore.

The ore comes from the mine more or less wet, and is first dried on a special drying kiln, and then crushed in a Dodge crusher. One man does the whole work for two Brückner cylinders, and tends to the ore-drying kilns, hauls his ore, takes care of the belting of the works, and looks after the screens, which must be constantly watched.

From the crusher the ore goes to the ball grinder, which is a cylinder with a rotary grate, the openings between the bars of which are less than $\frac{1}{64}$ in. In this cylinder 1000 lb. of iron balls, 3 in. in diameter, are made to revolve with the ore. When balls cannot be had, or when the stems of the stamp mills are so broken that they are fit for nothing else, they are cut to the length of their diameter and used in the place of balls. They soon become round from the motion of the cylinder. The balls wear out at the rate of 3 lb. to the ton of ore treated. The grinder needs very little repair and requires to be looked after only once a month. The crushed ore falls through the grate into a hopper, from the bottom of which it is carried by an endless chain to the upper floor, where it passes through a hexagonal revolving screen, covered with brass wire bolting cloth of 70 meshes to the linear inch. This cloth costs 90 cents a square foot.

This screen is 6 ft. long, the panels being 16 in. wide. One such screen is sufficient to do all the work of two Brückner cylinders. Flat screens are also used, which have a jogging motion, the ore being discharged from below them through a pipe. All that does not pass through these screens goes back to the ball grinder. It takes about four hours to run 3500 lb., or a charge for the cylinders, through the screens.

The object in crushing the ore so fine and bolting it is to get a pulp which will be of uniform consistency. The amount of water which is used in the barrel is only just enough to keep the pulp in a movable condition, and nct enough to dissolve the soluble salts, so that if there were any lumps in the roasted ore they would not become broken up or softened, and the silver in them would not be affected. Hence the absolute necessity of bolting out everything in the shape of lump and making the mixture with water in such a way that no lumps will form.

II. Roasting the Ore.

The bolted ore is discharged through a hopper into the cylinders. which have been fully described in the chapter on Roasting, The ore is always assayed after the charge has turned forty to fifty times in the cylinder. The time of roasting is generally thirteen to sixteen hours, and when the ore is exceptionally "heavy," it may be as high as twenty hours.

The roasted ore is discharged from the cylinders into wagons 6 ft. long, 2 ft. high, 30 in. at the top, and 29 in. at the bottom, with wheels 8 in. in diameter, placed so close together that it can be dumped. It is spread out on the floor to cool as soon as possible, in order to prevent the tendency to form lumps, and is then screened through a 40-to-the-inch screen. What remains on the screens are called screenings, which, together with the scrapings from the cylinders, are put on one side to be re-treated as described. The ore is now ready for amalgamation.

With five cylinders one man is constantly employed at the crusher. The labour account for these cylinders will be two roasters, two helpers, two crushers, one man on the cooling floor one amalgamator, and one helper. These are all first-class men except the helpers and cooling floor men, and are paid from $3 to $4. At night there are only three men in the mill, the roaster and his aid, and the crusher.

III. Amalgamation in Barrels.

From the cooling floor the roasted ore goes to the barrels to be amalgamated. There are five of these barrels, Figs. 121 and 122, in the mill, which are 4 ft. 6 in. in diameter, and 4 ft. 6 in. long. They are made with staves from 5 in. to 5½ in. wide, and 3½ in. to 6 in. thick, and are held together by four iron hoops, two at each end, and by an iron cross, the centre of which contains the journal, and the four end holes through which tie-rods pass, which are fastened with nuts. These iron cross-arms are placed on heavy cross-beams of wood which go over the head of the barrel. They cost, including the castings, about $200 each. When the staves are worn down to about 2 in., it is no longer safe to use them, and they are either replaced or lagged up on the inside, which process may be repeated indefinitely. The barrels are

made of pine, and usually last a year. Each barrel has openings in the centre of its length and at opposite diameters. Through

Fig. 121.

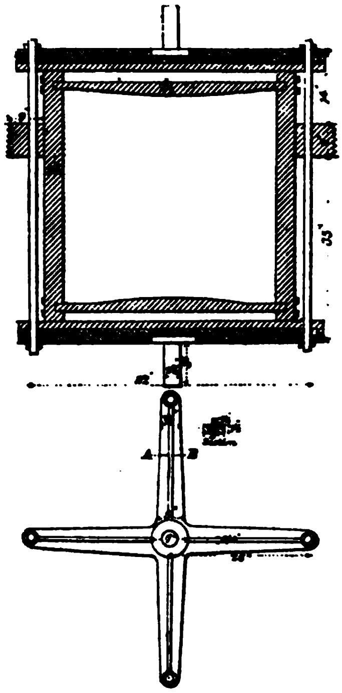


Fig. 122.

one of these which is the bung-hole proper and which is 5 in. in diameter, the charge is introduced. The other opening, which is opposite, is 1 in. in diameter, and serves for the introduction of the mercury.

The motion was formerly communicated to these barrels by means of a circle of gearing placed a little beyond the centre of the barrel, as it is in Europe. This has been abandoned and the common V gearing has been substituted in its stead. This is effected by means of a circle of wood, which is 6 in. wide and 9 in. high, set on end placed in the same position as the gearing. These V grooves are cut into this wood. Into these grooves similar ones on an iron wheel fit. Each barrel runs independently the one of the other. They were formerly thrown into gearing by means of a clutch, which was constantly wearing out, and would sometimes not bring the pinion into the gearing. So much

difficulty was found with it that the boxes in which the arbors run are now made eccentric, as shown in Fig. 123. By means of

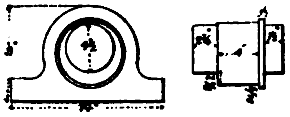


Fig. 123.

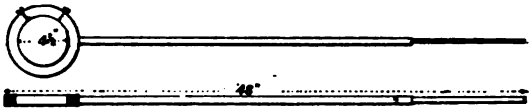


Fig. 124.

a crank, Fig. 124, which is fixed to the end of the arbor, any one of the barrels can be brought as near as is desirable to the fixed revolving iron V's. If they are in close contact they will, of course, revolve with a maximum rapidity. If they are at a little distance they will revolve slower, and in this way the barrels are made to revolve at any velocity, at will, or may be thrown out of gear altogether when it is desirable.

The power to run these shafts, as well as for the whole mill, comes from an overshot waterwheel 28 ft. in diameter and 10 ft. wide. This is much larger than there is any necessity for in ordinary times, but when the fork of Clear Creek, which furnishes the supply of water, runs low, it is necessary to have it of this size to run the works, as the amount of water is sometimes very small.

The barrel at the end of a previous operation, having been thoroughly washed with water, a charge of 2000 lb. of ore, and 60 lb. to 80 lb. of iron for light, and 150 lb. for heavy ores is introduced. The iron is almost exclusively worn-out mules' shoes, these being the cheapest kind of iron that can be procured. Water is immediately added to the charge, so that the ore will just pack in the hand but will easily fall to pieces. The bung is closed and the barrel is turned with this charge alone for four hours, at the rate of twelve to fourteen turns per minute. It is then stopped and the charge of mercury is introduced. This charge for an ore containing 150 oz. is 250 lb. A small quantity of sulphate of copper is introduced with the mercury. It is then

turned at the same rate of speed for sixteen hours. The barrel is then stopped to take an assay. The pulp is quite hot from the reactions which have taken place, so that when the bung is removed it makes a slight noise. When the bung is opened the assay is taken by introducing a stick like an ordinary lath to the bottom of the barrel and withdrawing it. When it is taken out some of the pulp will be attached to it, and this is carefully examined. If it has been properly worked the pulp will be about the thickness of dough and the mercury will be bright, and thoroughly incorporated with the mass. If the charge is too thin the mercury will have collected on the bottom, and there will be very little of it scattered through the pulp. Generally only one assay of this kind is needed when the character of the ore is known. Sometimes it will be found necessary to continue the operation and at others it will be found that the charge has been worked too long and that the mercury is floured and sometimes black. If the charge has been properly worked, water is introduced so that the barrel will be very nearly full. Previous to this time the charge had not more than half filled it. The barrel is set in revolution again for an hour at a slower motion, the object of which is to collect all the mercury and amalgam from the pulp on the bottom of the barrel and allow it to remain there. At the end of this time the barrel is turned over with the small opening down, an iron cock is inserted into it to draw off the mercury and amalgam which are on the bottom. In front of this hole there is a large trough in which an enamelled iron pot 12 in. in diameter and 6 in. high is placed. The mercury and amalgam are discharged into it and allowed to run until the charge commences to appear. After the amalgam is all out the stop-cock is withdrawn and the iron kettle removed, 100 lb. of fresh mercury are introduced, the opening closed, and the barrel made to revolve slowly for three-quarters of an hour. The object of this is to collect any mercury which may settle from the pulp, or which may have collected around the iron shoes in the barrel. The iron kettle is now replaced in the trough to receive the mercury again, after which the barrel is turned down with the bung toward the trough, and discharged into the sluices in which riffles are placed, and through which a rapid stream of water is constantly running.

The riffle bars are rectangular and are 2 in. in height. The mercury and amalgam which still remains in the pulp collects behind them. The trough containing the riffle bars ends in the dolly tub. Most of the mercury is caught in the riffles in the trough, but a part of it which is very fine is carried over into the tub. When the barrel is emptied it is filled one quarter full with water, and turned with the bung open until it is entirely empty. This will wash out some small quantity of mercury and ore, and as the water is likely to be thrown to some distance and must all be caught, the line of the revolution of the barrel, is protected on each side, with two vertical troughs, one 35 in. wide at the bottom and 5 in. deep, and the other 27 in. wide and 14 in. deep; both of them are placed close against the barrel and empty into the trough below into which the barrel is discharged. By this revolution with water everything in the interior of the barrel except the larger part of the iron shoes is washed out, and the barrel is cleaned ready for a new charge. With "heavy" ores about 3 lb. of iron per ton of ore treated will be used up in each operation, but with light ores much less. When the pulp is taken out of the barrel, the shoes which have been very much corroded come out in small pieces. These pieces are carefully scraped and put on one side wet. After they have slowly oxidized, the oxide of iron lifts the mercury which is then washed off. Only about 15 lb. of mercury is collected in the kettle which stands under the barrels and in the riffles, the rest goes into the dolly tub.

The mercury and amalgam from the barrel has been caught in the iron pot in the trough. These iron pots are used because it is not safe to carry a heavy weight in wooden vessels. They are the ordinary porcelain-lined kitchen utensils, and are much cheaper than any other vessel that can be had. From this vessel it is dipped into small ones. The amalgam will then be in two different pots. On the top of the last pot is a thick layer of pulp which is carefully removed, and washed in a wooden bucket. The amalgam is then washed with clear water, and the water taken off, the surface being very carefully dried with a sponge. After the amalgam has been in this way carefully cleaned, that which is taken from the riffles is added to it, and the whole is then weighed. That which comes from the dolly tub, when it is

cleaned, is weighed separately and then added to the rest. The whole is now placed in a bag of duck to be strained.

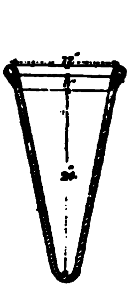


Fig. 125.

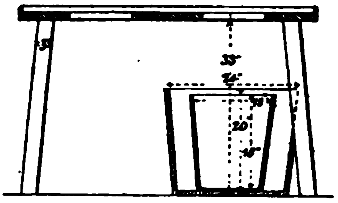


Fig. 126.

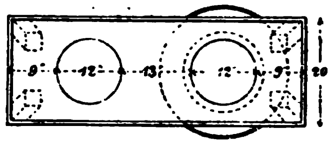


Fig. 127.

This bag, Fig. 125, is conical, 12 in. in diameter at the top, and 2 ft. long. The top is made of ¾-in. round iron, bent in the form of a circle 12 in. in diameter on the inside, over which leather is so folded as to be double and to fall 3 in. below the ring; it is stitched directly beneath the ring. Between the two open folds at the bottom the canvas bag is fastened. The iron ring of this bag will just fit over one of two holes, 12 in. in diameter, made in the amalgamation table, Figs. 126 and 127. This table is 4 ft. 6 in. long, 1 ft. 8 in. wide, and 33 in. high. The top is made of 1 in. plank, and is provided with a rim ½ in. in height in order to catch any mercury that may fall upon the top. Below the bag hole is a wooden tub 2 ft. in diameter at the top and 2 ft. high, the bottom of which is covered with water. In it an iron vessel is placed which is 15 in. in diameter and 18 in. high, into which the canvas bag projects. The object of having two tubs, one inside the other, is to prevent any possible loss of mercury or amalgam.

The clean mercury and amalgam is brought in the iron pots and poured in the canvas bag, which is rarely ever allowed to be more than half full at a time. The excess of mercury runs immediately through the pores of the cloth, and when the amalgam is so thick that no mercury runs, the bag is twisted as tight as possible with both hands, and when no more can be got out by twisting, the bag, twisted as it is, is lifted out on the top of the table and squeezed and rolled, and when no more mercury can be got out in this way it is ready for the furnace. After the

amalgam is thoroughly strained, it is weighed, and the difference between it and the weight of the mercury and amalgam gives the loss in the barrel. The loss is very large, generally from $2\frac{1}{2}$ lb. to 6 lb. per ton of ore treated.

When the amalgam is pure it has a crispy, somewhat sandy feel, and is more or less hard, and makes a slight noise, somewhat like the rumpling of silk when pressed in the hand. It does not, however, have a cry like that of tin. It is more or less hard, except when lead is present in considerable quantities, when it is soft. The fineness of the bullion depends almost entirely upon the way in which the roasting is done. If the roasting has not been properly performed, the bullion can only be made fine at the expense of the mercury.

The Dolly Tub.—The dolly tub is of wood 5 ft. in diameter and 6 ft. high. A stirrer of wood revolves in it. The vertical shaft has four arms. Each arm is bound by a brace to the one next to it, and the set of four braces make a square, the arms being the diagonals. Each one of these braces carries a long pin, and the four arms are fitted with them. These pins go to within an inch of the bottom of the tub. The shaft is run at the rate of twenty-five revolutions per minute. The tub is kept constantly full of water, and receives the whole charge from the barrels. It has in the sides a number of holes, the lower one of which is 4 in. from the bottom, for discharging the contents of the tub when it is to be cleaned up to get the mercury, which is once a week, or once in two weeks, as occasion may require. A stream of water constantly runs into the tub. From the upper hole the water flows continuously, carrying off all the light material into the stream. All the heavier materials collect in the bottom, and are always assayed. If the charge in the barrel has been properly worked, the pulp caught in the bottom of the dolly tub is thrown away, if not it is put on one side to be re-treated. It consists of base metal, oxides, gangue, and silica. In order to collect it, it is simply washed out into a reservoir, allowed to settle, and then dried. It is then roasted in the cylinder with raw ore and salt. There were at one time 16 tons of tailings retreated in the course of three months, which averaged from 35 oz. to 45 oz. of silver. This was caused partly by the ignorance and inexperience of the roaster and amalgamator, but mostly

from neglect in assaying the charges, and is entirely unnecessary. One dolly tub is sufficient to do the work of two barrels.

IV. Distilling the Amalgam.

The dry amalgam is placed in four half-round cups which are made to fit the bottom of the cast-iron retort. This cast-iron retort is 30 in. long, 14 in. in diameter in the front and 9 in. at the back, having a recess in the front in which to place and fasten the iron cover. It is placed in a furnace of ordinary red brick, which is 53 in. high, 46 in. wide, and 34 in. deep, the whole being tied in both directions with iron braces 1½ in. wide. The top of the furnace is an arch one brick thick and 21 in. in diameter, the pillars which support it being 12 in. wide. The retort is supported at both ends by pieces of refuse stamp stems, as those are usually the cheapest material to be had at the works. There are twelve grate bars ¾ in. square, placed 8 in. from the ground, and 14 in. from the retort. The small end of the retort usually projects through the brickwork, and from the centre of the end, cast in one piece with it, is a pipe 2½ in. in diameter outside, with a flange at the end. Into this pipe a 2-in. iron gas-pipe is screwed, so as to be gas-tight, and runs a few feet beyond the body of the furnace, slightly inclined, and then turns at right angles and passes to the bottom of an ordinary barrel filled with water, then through the sides of this barrel on a slight incline over a tub partially filled with water into which the condensed mercury drips. The object of the water in the tub is to prevent the mercury from spattering as it falls. The four cups hold together 450 lb. of amalgam. When placed in the furnace the door is put into the front of the retort and made gas-tight, so that no fumes of mercury can possibly escape from it, and the fire is lit on the grate. It takes a slow fire four to five hours to completely distil this charge. If the angle of the pipes does not incline sufficiently some mercury is caught in them, and sometimes is considered as lost. It may be recovered, however, by gently tapping the pipes, and is always regained in subsequent operations. When the charge is finished the furnace is left to cool slowly and the amalgam is removed from the cups, which are carefully

scraped. It is then weighed. The difference between the weight of the amalgam and that of the retort silver and distilled mercury, will give the loss in mercury. Theoretically there should be no loss, but there is always a slight difference between the weights, owing to the fact that the furnace is not always perfectly tight and that some amalgam is either actually lost or is counted as lost.

The following Tables give several samples of the treatment of Pelican and purchased ore, and of the results in melting retort silver:

Amount of Work Done in the Mill July 6th, 1874. *One Day's Run on Sweepings, Dirt, and Scrapings, from Purchased Ore. Barrel No.* 1. *Charge No.* 210.

Charge of ore ...		2000 lb.
Silver contained		66 oz.
Time of charging		10 a.m.
Mercury charged		110 lb.
Time of charging mercury		2 p.m.
Mercury and amalgam discharged		115½ lb.
Amount of amalgam ...		40 lb.
Weight of retort silver		90 oz.
Fineness of the bullion		800 to 900
Weight of retort silver	90 oz.	
Loss in melting	5 ,,	
Weight of melted silver		85 oz.
Yield of silver 800 fine, 800 × 85 = 68		
Loss in melting ... 1½ lb.		

The base metal was copper.

One Day's Run on Pelican Ore. Barrel No. 2, *Charge* 211.

Charge of ore ...		2000 lb.
Silver contained		134 oz.
Time of charging		10 a.m.
Mercury charged		220 lb.
Time of charging mercury		2 p.m.
Mercury and amalgam discharged		234½ lb.
Amalgam		84 lb.
Weight of retort silver		175 oz.
Fineness		800
Weight of retort silver	175 oz.	
Loss in melting	9 ,,	
Weight of melted silver		166 oz.
Fineness 800, 166 × 800 = 132.8 oz.		

The base metal was lead. The amalgam generally retorts 2 oz. to 2.15 oz. per pound.

Fusion.

Sample of Selected Ore Purchased.

Barrel No. 25, Monday, June 15th, 1874.

	oz.	oz.
Barrel No. 1 contained	191	
,, 2 ,, ...	187	
	378	retorted 387
Discharged June 16th.		
Barrel No. 1 contained	174	
,, 2 ,, ...	166	
	340	retorted 385
Wednesday, June 17th.		
Barrel No. 1 contained	166	
,, 2 ,, ...	169	
	335	retorted 431
	1053	1203

	oz.	
The weight of retort silver	1203	
Weight of the bar	1141	
Remains for the next bar	62	
Weight of fine silver in the bar at 832 fine ...	...	949.3
Weight of fine silver in 62 oz. at 832 fine ...	...	52.4
Total fine silver ...	...	1001.7
Total fine silver by assay	1053	
,, ,, found	1001.7	
Total loss	51.3	
Yield in fine silver	...	95.3 per ct.

	oz.
Bar 24 gave	95.1
,, 25 ,, ...	95.3
,, 26 ,, ...	96.
The average yield in purchased ore is	93 per ct.
,, ,, on Pelican ore is	87 ,,

It was impossible to get any economic data with regard to the treatment at this mill, owing to the fact that the work was done for the most part for the owners of the Pelican Mine; this mine having been in continued litigation with the Dives Mine, and being subject to constant visits from legal officers, has frequently

been obliged to work with sentinels posted so as to give notice of their approach.

The cost* of this process at the Mettacom Mill in 1869 was:—

ACTUAL COST OF POWER.

	$
2 engineers, \$6 and \$5.50 per day	11.50
4¾ cords of wood at \$10	47.50
Repairs and oil	5.00
Cost of power per day	64.00

COST OF CRUSHING.

	$
Power (batteries require about half the power) ...	32.00
2 men to break rock and feed, \$4.50 and \$4.00 ...	8.50
1 man to clean battery and help them	4.00
Average daily repairs	3.00
Total daily expense of batteries	47.50
Average daily crushing, 13 tons; cost per ton ...	3.65

COST OF ROASTING.

	$
11 men at \$4.00	44.00
1 foreman at \$5.50	5.50
4 cords of wood at \$10.00	40.00
Salt, 10 per cent. on 6½ tons, 0.65 tons at \$45.00 ...	29.25
1 cooler (also employed at the barrels, charge half time) at \$4.00	2.00
Repairs and incidentals	8.00
Total daily expense of furnaces	128.75
Average roasting per day 6½ tons; cost per ton... ...	19.80

COST OF AMALGAMATION.

	$
Power (half the power of the engine)	32.00
Old wrought iron, 100 lb. daily, at \$2.50	2.50
3½ men at \$4.00	14.00
Loss of quicksilver, 2 lb. per ton	13.20
Repairs	2.00
Light and oil	2.00
Total daily expense of barrels	65.70
Average daily amalgamated 10 tons; cost per ton ...	6.57

		$
Cost of crushing	per ton	3.65
,, roasting	,,	19.80
,, amalgamating	,,	6.57
Retorting and melting	,,	1.00
Total cost of treatment	,,	31.02

* Mining Commissioners' Report, 1870, p. 739.

COST OF TREATMENT OF CHLORIDE ORE.

	$
Crushing, about	3.50
Amalgamating	6.57
Retorting and melting	.75
Total	10.82

The power for driving stamps, barrels, settlers, &c., is furnished by an engine of about 60 horse-power run at 40 to 45; steam pressure 55 lb. to 60 lb. Steam is supplied by two 16-ft. tubular boilers, 44 in. in diameter, and containing 42 tubes each. The steam cylinder has 14 in. diameter and 30 in. stroke. There is an 18-ft. flywheel, weighing 4500 lb.

The work by barrel amalgamation should be just as well done as by the pan, though it is much slower. The pan, however, is peculiarly an American process, and is the one which has grown up from the necessities of the country, and is, therefore, the one which is best known, while the barrel amalgamation is a foreign process, and one against which there is a great deal of prejudice. It is evident that the plant of a works with barrels, must be very much cheaper than it possibly can be with the pan process. The difference in labour is not very great. The yield is about the same, and barrel amalgamation should therefore be somewhat cheaper than pan. It has, however, never come into any extended use, probably because it cannot be said to have had a strictly fair trial at any mill where it has been introduced. It has been frequently abandoned for the pan for the very obvious reason that the work of the pan is thoroughly understood, while that of the barrel must generally be first learned by the engineer in charge and taught to the men. The pan gives a much greater output in the twenty-four hours, and it works quicker, and takes up but little more room than the barrel. It is open on every side, and is generally drier to work with. It is therefore not suprising that the pan should be preferred and used. In a given case the pan would probably be the cheapest for any company, as pans of every variety may be had at any foundry for a comparatively small price, as they are obliged to keep all the patterns on hand, beside which the skilled labour necessary to superintend the process and to teach it to green hands, can always be had, which is not the case with the barrel.

CHAPTER VIII.

PAN AMALGAMATION.

PAN amalgamation is the process most extensively used in the West for the extraction of silver from its ores. The works where the process is carried out are called mills; leaching plants are often connected with them for the treatment of tails. Two processes are used which differ from each other according to the character of the ore. They are known as the Washoe and the Rees River processes, from the districts in Nevada where they originated. They differ very slightly. In the Washoe process the ores are of such a character that they require only to be stamped, and then go directly to the mill to be amalgamated; while in the Rees River process, if crushed wet, they require to be dried on drying floors, heated by the waste heat or by furnaces, and then roasted, previous to being amalgamated. The use of drying floors is being rapidly given up. Their place is taken by kilns or rotary furnaces, which utilise the waste heat equally well, are more economical in labour, and effect the drying more completely and in less time than the floors. The processes of roasting vary in different localities at different times. There are a number of furnaces which have been invented for this purpose, which have been more or less successful, most of which have been described in the chapter on Roasting. They are generally reverberatory furnaces with a number of hearths, either placed the one after the other, so as to make one long furnace, or built at different levels, sometimes directly over each other, or in the shape of steps. The only shaft furnace which has continued in use is the Stetefeldt furnace, which is applicable mostly to "light" ores, or those containing very little sulphur. Of revolving furnaces, Brückner's cylinder, and the other furnaces which are almost identical with it, are extensively used.

In discussing these processes, no particular mill will be described, since they all resemble each other. The details for pan

amalgamation which are given have been taken for dry crushing from Stewart's, and Judd and Crosby's Mills in Georgetown, and the Nederland Mill in Boulder, Colorado; for wet crushing, from the Brunswick, Eureka, and Consolidated Virginia Mills in Nevada, the Ontario Mill in Utah, the Lexington Mill in Montana, and the Tombstone Mill in Arizona. Some of these mills have ceased to exist, others do not now have the importance which they once had, but the work done at one time by most of them was in some respects typical, and the results obtained are valuable either as history, or as real acquisition, to the sum of our knowledge of these subjects. The processes for crushing ore have been described in Chapter IV. Barrel amalgamation, as practised in the Pelican Mill in Georgetown, Colorado, has been described in the previous chapter.

The general arrangement of the mills is the same, exception being made of the presence or absence of roasting. When roasting is done, the ore is crushed dry; when it is not, it is crushed wet. About 25 to 30 per cent. of the Comstock ores is gold, the rest is silver; but in the bullion produced the proportion is somewhat higher, as the gold is more completely saved than the silver. In other mills, like the Tombstone, the amount of gold is smaller and the quantity of it saved much less. As a general thing, the richest ores are sulphides, and must be treated by dry crushing and roasting, and the poorer ores, which are oftener free milling, are crushed wet, and directly treated by amalgamation, which is the Washoe process. The ores treated in 1874 at Stewart's "custom" mill in Georgetown, Colorado, averaged 150 oz. of gold, 5 per cent. of lead, and 30 to 73 per cent. of zinc, mostly zincblende. There is a trace of copper and some gold, but not enough to separate it. The rock is quartz, and contains from 30 to 50 per cent. of ore. Judd and Crosby's "custom" mill treated ores of about the same grade. Both these mills roasted in reverberatory furnaces. The Nederland Mill was a "company" mill, and treated ores from its own mines. They contained 50 oz. of silver, 3 per cent. of lead, 2 per cent. of copper, and 63 per cent. of quartz. The roasting was done in Brückner's cylinders. The Ontario and Lexington Mill in Nevada use the Stetefeldt furnace.

In all the large mills the ore is assayed by the assayer of the company. The samples are taken from the mine or from the ore delivered. In the speculative mines this is a matter of great secrecy, and only the superintendent and chief officers of the mill know its history. The sample is wrapped in wrappers which are printed like the bullion assay wrappers, both on the inside and outside, so that there shall be no mistake. Three samples are taken; two go to the assay office, and the third is kept in the office of the mill. A careful record is kept in both places of all the numbers and marks on the papers as well as of the assays, which are reported on the blank given at the end of this chapter. At the end of a time determined, when it is certain that there will be no call for further assays, all the assay samples are collected and added to a charge in the pan.

In Colorado, in 1874, 40 oz. of silver could be treated with profit by the mills, but only 60 oz. yielded a profit to the miner; 20 oz. about paid the expense of the process. The ore treated at the Eureka Mill in Nevada yields from $50 to $160 of gold and silver, three-fourths being gold and one-fourth silver. For the prices paid in 1885 for Colorado ores, see p. 76, and the description of the Boston Colorado Works, p. 119. Below, the prices of two different works as they were advertised in July, 1874, are given:

PRICES GIVEN FOR SILVER, GEORGETOWN.

Ounces of Silver per Ton.	Price per Ounce of Silver for Ores containing from 5 to 10 per Cent. of Zn.	Price per Ounce of Silver for Ores free from Zn or containing less than 5 per Cent.
	cents.	cents.
75 to 100	48	50
100 ,, 125	55	60
125 ,, 150	65	70
150 ,, 175	75	80
175 ,, 200	82	84
200 ,, 250	85	88
250 ,, 300	90	92
300 ,, 350	93	94
350 ,, 400	96	98
400 ,, 450	99	100
450 ,, 500	102	103
500 ,, 550	104	105
550 ,, 600	106	107
600 ,, 700	107	108
700 ,, 900	...	109
1000 and upward	...	110

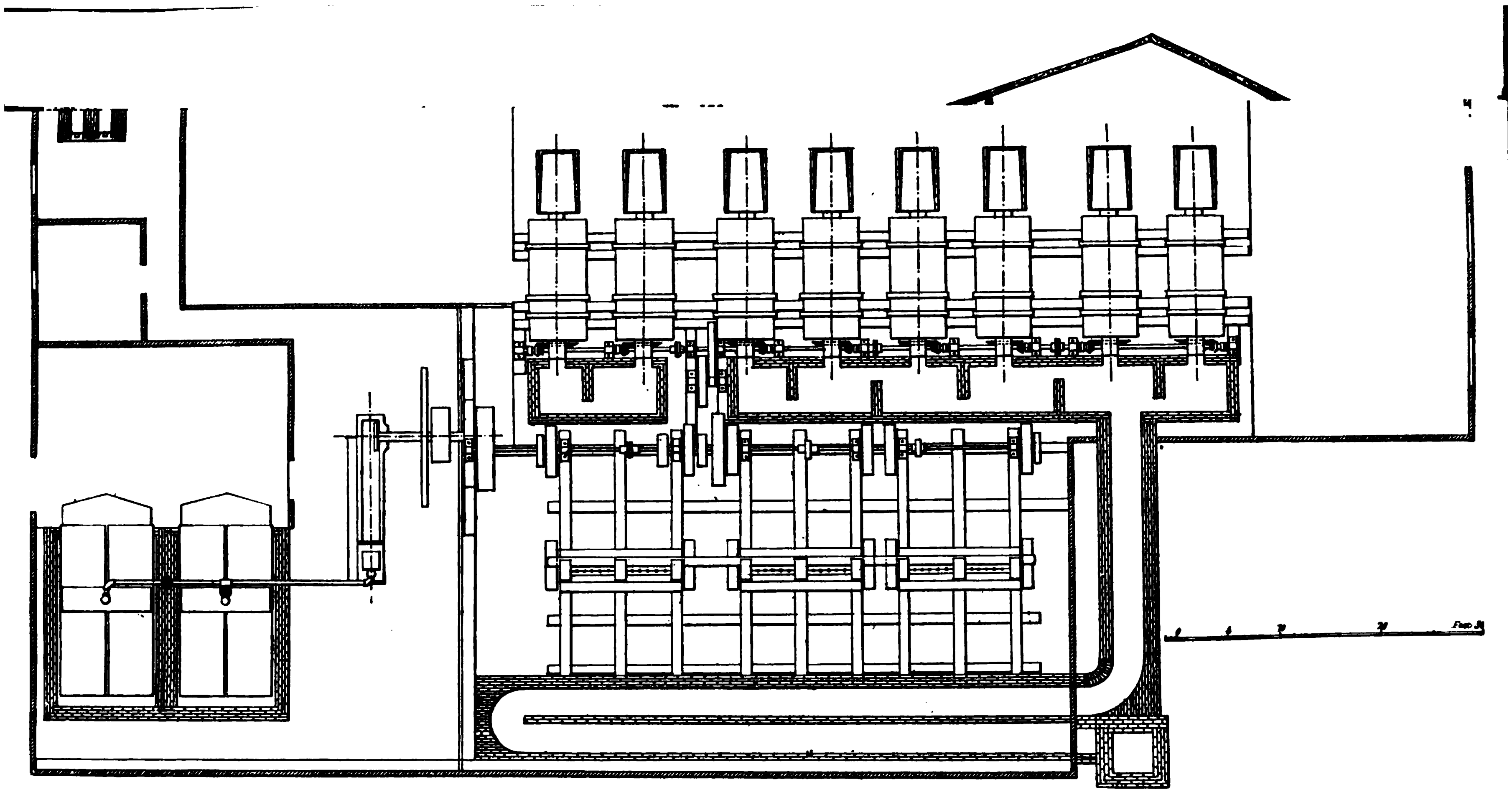
At Stewart's Mill, Georgetown, the following prices were paid:

		oz.			cents.		
Ore	containing	299	silver	or less,	105	for each oz.	over 35 oz.
					No deduction.		
,,	,,	300	,,	,,	94	,,	,,
,,	,,	325	,,	,,	95	,,	,,
,,	,,	350	,,	,,	96	,,	,,
,,	,,	375	,,	,,	97	,,	,,
,,	,,	400	,,	,,	98	,,	,,
,,	,,	450	,,	,,	99	,,	,,
,,	,,	500	,,	,,	100	,,	,,
,,	,,	550	,,	,,	101	,,	,,
,,	,,	600	,,	,,	102	,,	,,
,,	,,	650	,,	,,	103	,,	,,
,,	,,	700	,,	,,	104	,,	,,
,,	,,	750	,,	,,	105	,,	,,
,,	,,	800	,,	,,	106	,,	,,
,,	,,	850	,,	,,	107	,,	,,
,,	,,	900	,,	,,	108	,,	,,
,,	,,	1000	,,	,,	110	,,	,,
,,	,,	2000	,,	,,	112	,,	,,

The ore is delivered to rock breakers, placed at the highest level of the mill. Below the rock breakers, between them and the batteries, or at each extremity of the battery line and a little behind it, the drying floor in dry crushing is placed. This floor in all the older mills is made of cast-iron plates, 36 in. by 42 in., flanged on the sides, so that the plates may overlap and still give an even floor. A flue from the roasting furnace runs backwards and forwards under this floor, the partition walls of which serve as supports for the plates. This floor is placed directly behind the stamps, and the ore is spread over it until dry, when it is charged by hand or automatically into the stamps, which are on the same level. This system is gradually giving place to regular furnaces or kilns for drying the ore, in which the work is much more satisfactorily and economically done. At the Ontario Mill a revolving dryer is used, and at the Lexington Mill the shelf dry kiln.*

Below the discharge level of the stamp, the roasting apparatus, if the ore is to be roasted, is placed in such a position that the ore from the stamps can be discharged into them, or be carried by a screw conveyor or an endless chain to an elevator which discharges into hoppers containing a charge for the furnace, when a revolving furnace is used, or in the case of the Stetefeldt furnace into the mechanical charging apparatus when the ore is

* Trans. Am. Inst. Min. Eng., vol. xii., p. 95.



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DRY CRUSHING SILVER MILL, CONSTRUCTED IN 1873 BY THE UNION IRON WORKS OF SAN FRANCISCO, CAL.

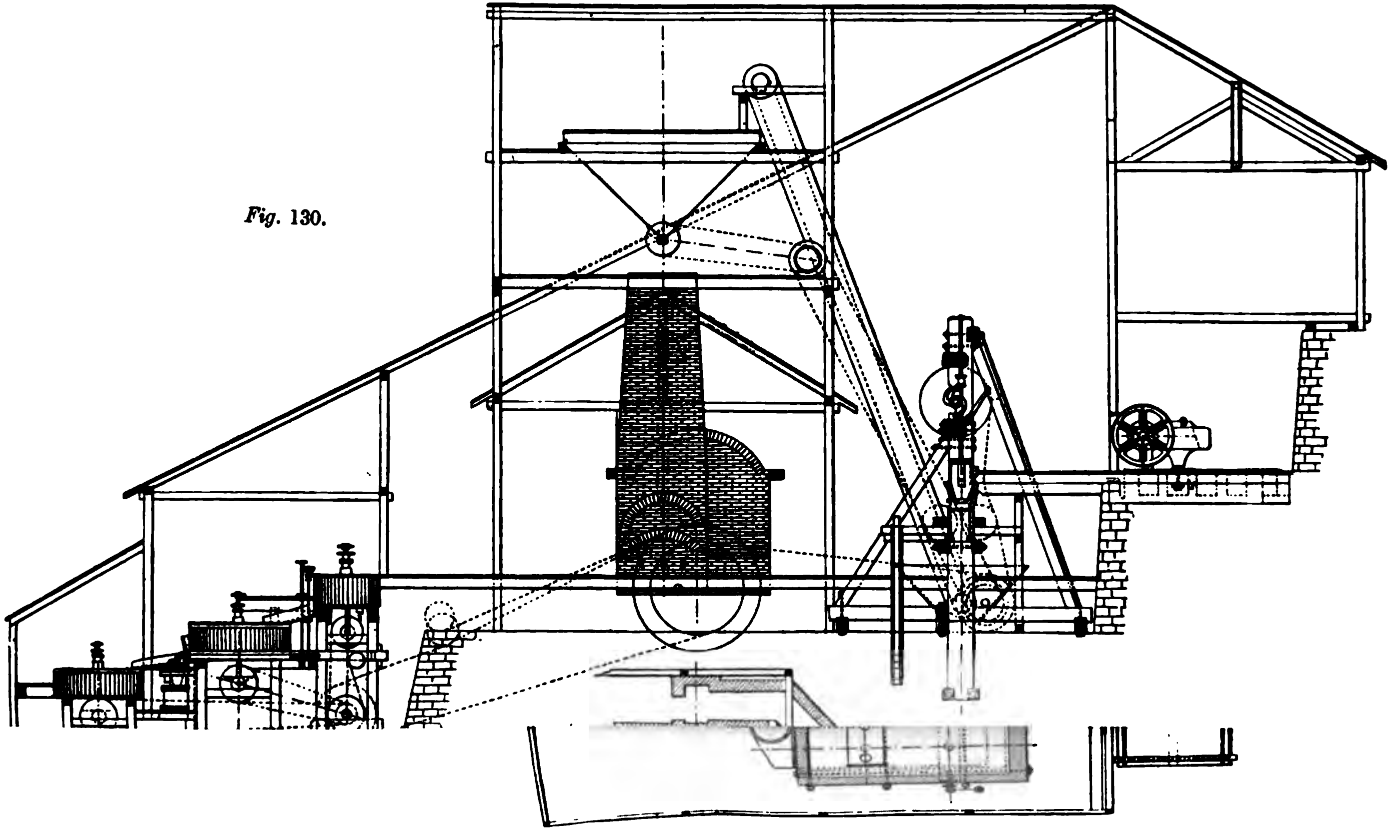

Fig. 130.

To face page 353.

to be roasted or chlorurised, or both. In all the older mills the roasting furnace was placed directly in front of the batteries. In the best modern ones they are placed at the extremity of battery line and a little behind it. The supply of ore is cut off from the furnaces automatically, when the hoppers are full or the charge is continuous, as in the Stetefeldt furnace. The speed of the elevator feeding the pulp into the hoppers is regulated by two sets of cone pulleys, and may be made to vary from 40 to 300 revolutions per minute. In front of the furnace there is a brick or iron cooling floor, where the ore remains until it is cool enough to be charged into the pans. On a still lower level are the pans and catch pits, if the mill is wet crushing, and lower still the settlers.

The arrangement of the mills as they were generally built previous to 1873 is shown for dry crushing with the Brückner cylinders in the plan of the Nederland Mill of Boulder, Colorado, in Figs. 128 and 129, and with the Stetefeldt furnace in Fig. 130. Since that time the use of drying floors has generally been abandoned for some kind of mechanical drying furnace, as being at the same time healthier for the men and cheaper to manage. It was found that, as the floors were not tight, fumes of arsenic and other deleterious substances from the ores escaped through the cracks and affected the health of the men, besides which the soles of their feet being constantly warmed enervated them and rendered them liable to sickness from this cause. The disposition of the mills with the Stetefeldt furnace has also undergone considerable change. The hopper shown in Fig. 130 is no longer used, but in its place the charging apparatus shown in Fig. 102, which is supplied by means of a chain with buckets or some other automatic appliance. The furnaces, instead of being placed in front of the stamps and between them and the pans, are now placed in the best constructed mills at each end and a little behind the line of the stamps. The mechanical drying furnaces for the ore and salt are placed directly behind the stamps and in the same line with the roasting furnaces. The most recent disposition of dry crushing mills with the Stetefeldt furnace is well illustrated by the arrangement of the Lexington Mill of Butte City, Montana, Fig. 132, which is a

50-stamp mill. It was built in the year 1882, and is one of the best of the modern mills. It is situated on the side of a hill, the difference in level between the top and the bottom being 61 ft. The ore is brought to the highest level and is dumped on grizzlies and passes to the Blake crushers. It then goes to the shelf drying kilns directly back of the stamps. The salt to be used with the ore is dried in the same kind of a kiln, which is capable of drying six to seven tons in a day. The stamps weigh 850 lb. There are forty for ore and ten for salt. They are fed by a Tullock feeder, one for each battery of five stamps. The fall of the stamp is 7 in. The mortars discharge on both sides through 30-mesh screens. The crushed ore is conveyed by a screw to a point where it is raised in buckets to screens. What does not pass, or about one-tenth of the total stamped ore, is returned to the stamp. The salt is carried in the same way. The salt and ore are mixed mechanically after passing the drying furnace and are elevated to the top of the roasting furnace. The Stetefeldt furnace is on a line with the dryers but outside of them and at opposite extremities of this part of the mill. The cooling floors in front of them are very large and convenient. In front of the cooling floor there is a depressed railroad on which the wagons for salt and ore are placed, so that they are shovelled directly into the top of the wagon from the bottom of the floor. From here they pass by rail to the pan floor, which is on the level of the rails. The roasted ore is thus discharged into the pans with the greatest ease and with the least amount of handling. In order to prevent any action of the dust on the stamping machinery, the mortars of the stamps are closed with wooden boxes, to which exhaust fans are attached to draw off the dust into pockets, which are emptied once a week. The line of pans, of which there are twenty, is parallel with the stamps. There are ten settlers on a lower line. A Corliss engine of 240 horse-power runs the mill. The mill treats 250 tons of ore in twenty-four hours and uses twenty-two to twenty-three cords of wood.

The power for the mill is generally steam, sometimes water, and sometimes both water and steam together, the steam being used when the water fails. The power is communicated to the

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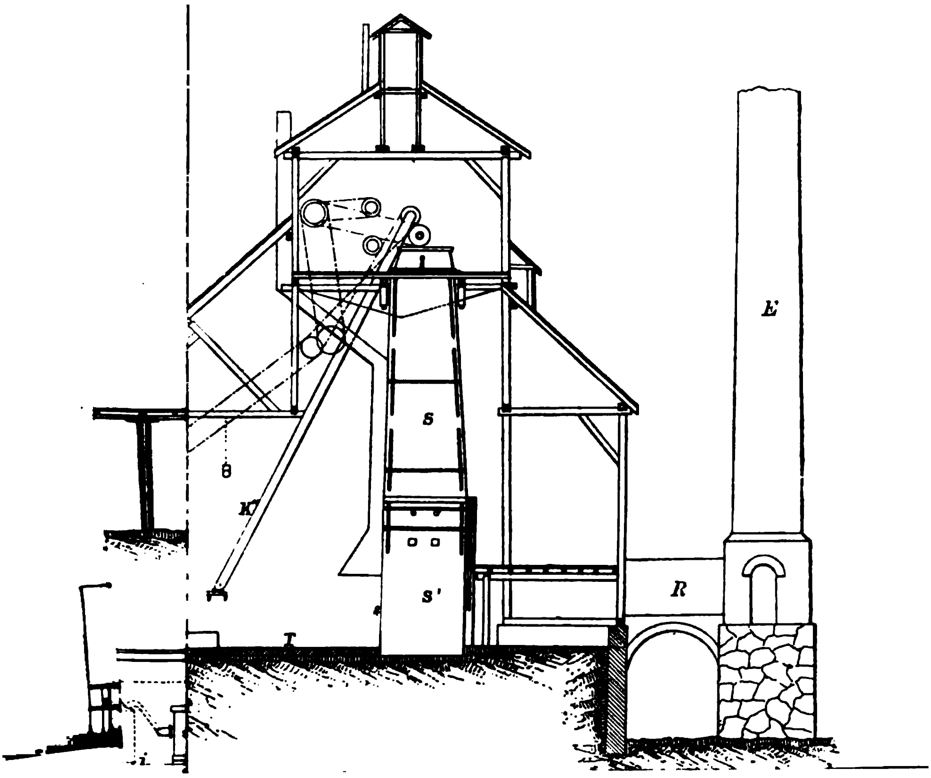


various parts of the mill by shafts above and underneath the floors. Each kind of machinery has its own independent line of shafting. The power is always transmitted by belting. The stamps are so arranged that any stamp or any battery may be hung up in a few moments without stopping the mill, and each pan can be detached from the others and remain motionless. The power required to drive the pan is from three to five-horse power, according to its capacity. The power required to crush and amalgamate one ton of ore will vary from four and a half to six and a half horse-power, depending upon the way the mill is managed.

The process of amalgamation is in general the same in all the mills, the barrel process having almost entirely disappeared. The process of crushing is generally the same, and consists in the use of revolving stamps, the crushing being either dry or wet according to circumstances. Silver ores, which can be amalgamated directly, are generally crushed wet; those that are to be roasted are crushed dry. The general outline of crushing is, first, to reduce the rock in a Dodge,* Fig. 138, or Blake's crusher, Fig. 139, to a certain size. It is then fed under stamps, arranged in batteries of five, the weight of which will vary from 600 lb. to 1000 lb.

If the mill companies treat their own ores only, the mills are said to be "company" mills. If they do not, but treat ores for others, the mills are said to be "custom" mills; the only difference being that in the first case the ore is dumped into shoots, and treated as it comes to the mill, while in the latter case it is kept carefully separated in bins made for the purpose, and each lot treated by itself.

The ore, whether it is purchased, treated for others, or from the mines of the company, is sampled as it comes from the mine. This sampling is usually done on the crushed ore, the sample being taken as it falls through the crusher. Some of the mechanical methods of making assays have already been described.†

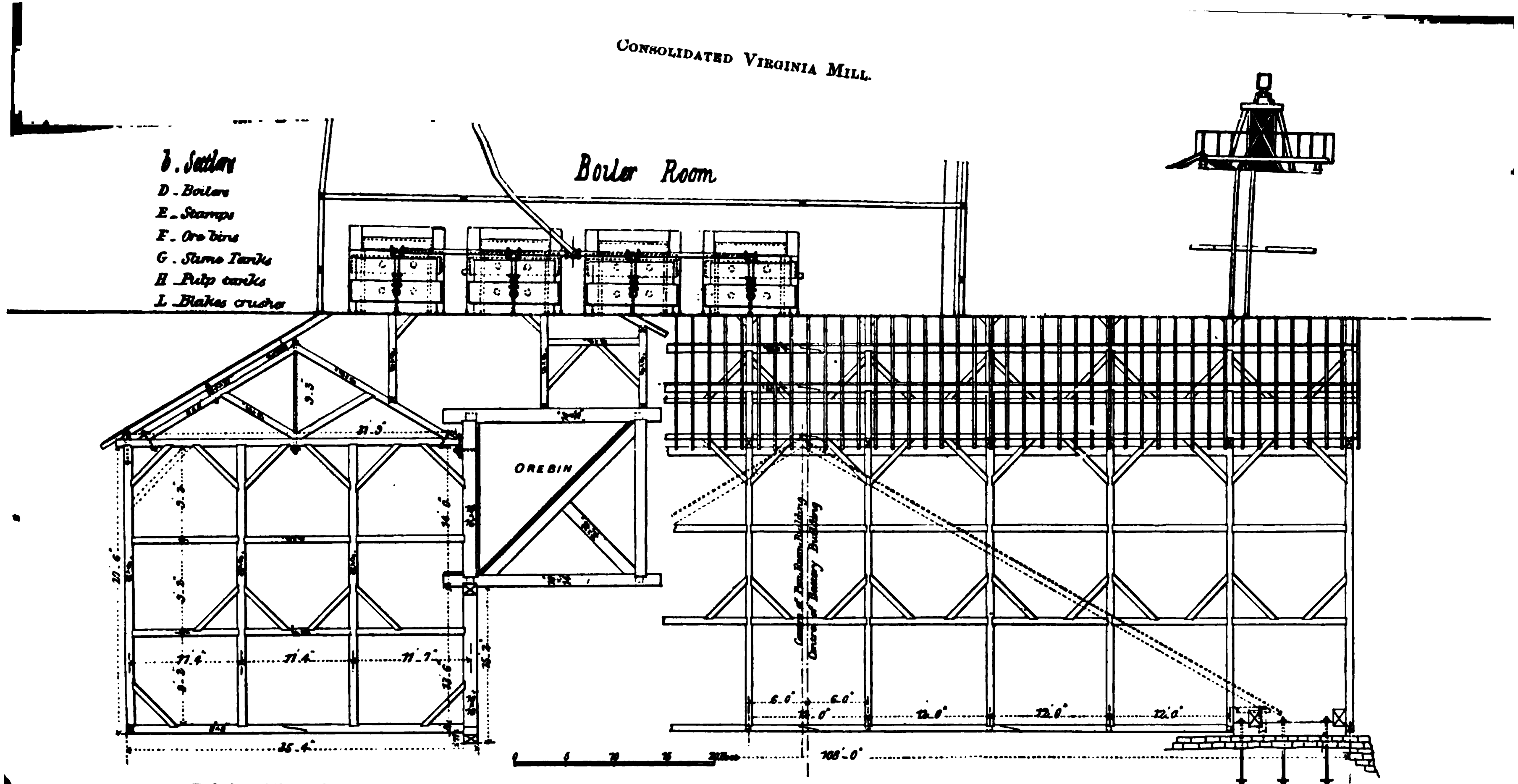
* The Dodge crusher is used very largely in most small mills, because the ore can be easily sampled with it. In all the large mills the Blake crusher is used.

† Page 66.

At Stewart's Mill the assay sample is taken mechanically by a very simple arrangement. The ore from the jaws of the crusher falls down a shoot into a conduit passing to the elevator. Below the shoot, in the conduit, a box about 11 in. long and 2 in. wide, is fixed to an iron rod which makes twelve revolutions a minute. At every revolution of the rod, the box passes under the mouth of the shoot. It is filled with ore, and empties itself through an opening in the conduit into a box beneath it. The machine is calculated to collect a sample of 75 lb. to the ton of ore. In this same mill, in addition to the crusher sample, the ore is also automatically sampled for the guidance of the mill managers on the battery pulp. The pulp is discharged from the stamps through screens with fifty meshes to the inch upon the mortar apron, which has three holes $\frac{3}{8}$ in. in diameter, through which a certain portion of the ore drops into the same arrangement which is used in the crusher. The sample taken is about 70 lb. from every ton, each lot of ore being always kept separate. A sample from this is taken in the ordinary way. As the discharge from the stamps is uniform, the sample so collected will represent the average value of the ore.

The ore is usually delivered by rail into the ore bins at the highest elevation of the mill, and from this point it must descend without handling. Figs. 133 and 137 show the arrangement of the ore bins at the Consolidated Virginia Mill. It is usual to place the rock breaker directly over the ore bins, which should be constructed at such an angle that the ore will slide from them towards the stamp. They should be large enough to hold at least several days' supply of ore, so that the mill, in case of a full supply of ore is not on hand, may not be shut down on that account. The ore is usually screened though not always. The object of screening is not only to size the ore, but also to separate the pieces of iron tools which are frequently found in it, and which would damage the crushers and stamps if they passed through them. When the ore is not screened, the largest pieces are picked out by hand to go to a crusher, which is generally Blake's. Fig. 138 shows a section of one of these crushers, of which four different varieties are made. It is the most extensively used of all the crushers. The amount that it can crush in a given time depends

CONSOLIDATED VIRGINIA MILL.



End view of the gable end.

Lower side elevation of the building. The pan room roof is shown in dotted lines.

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on the distance between the jaws, and the velocity, both of which can be regulated to suit any kind of rock or size of crushing.

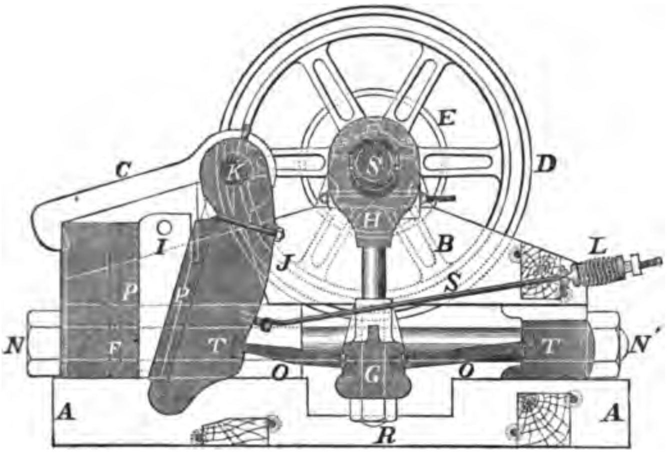


Fig. 138.

A. A.	Lower timber frame.	J.	Swing jaw.
B.	Upper ,, ,,	K.	Jaw shaft.
C.	Clamps	L.	Spring on spring rod.
D.	Fly wheels.	N. N.	Main tension rod nuts.
E.	Pulley.	O. O.	Toggles.
F.	Main frame.	P. P.	Jaw plates.
G.	Brushes.	R. H.	Pitman.
H.	Pitman half box.	R.	Pitman rod nuts.
I.	Chucks.	S.	Main eccentric shaft.

The capacity given in the Table below supposes that the jaws are set to open 1½ in. at the bottom, and that the crusher is properly fed and is running at a speed adapted to the rock being crushed, as hard stone which breaks with a snap, will be much more rapidly broken than tough rock. It can generally be calculated, when it is desirable to know the weight of ore being crushed, that a cubic yard of rock will weigh about 1⅓ to 1½ tons. Generally the castings are made of one piece; but when transportation is difficult, the crusher is often made in sections and bolted together, no single piece weighing more than 350 lb. In setting up a crusher it is generally desirable to have a little more power than that just necessary to drive it. The following Table gives the sizes and capacity of the crusher.

Size, or receiving Capacity.	Product per Hour in cubic yards.	Weight of Heaviest Piece.	Weight of Timber Frame.	Total Weight.	Extreme Dimensions. Length.		Extreme Dimensions. Breadth.		Driving Pulley. Diam.	Driving Pulley. Face.	Proper Speed.	Horse-power required	REMARKS.
					ft.	in.	ft.	in.	in.	in.			
$3 \times 1\frac{1}{4}$	—	—	—	125	1	8	1	0	—	—	—	—	For laboratory use and sampling.
5 × 4	one	130	55	750	3	0	1	3	13	4	300	$1\frac{1}{2}$	Takes pieces 4 in. in diameter; crushes to corn size.
6 × 2	—	130	55	750	3	0	1	3	13	4	300	$1\frac{1}{2}$	Takes pieces the size of an egg; crushes to corn size and dust.
15 × 2	—	295	325	2,690	4	6	4	0	16	5	300	3	Takes pieces the size of an egg; crushes to corn size and dust.
10 × 4	three	300	375	2,850	5	6	3	9	16	5	300	4	Takes pieces 4 in. thick; crushes to $\frac{3}{4}$ in. or less.
60 × 2	—	840	600	6,000	7	0	5	6	20	10	275	9	Takes pieces 10 in. by 7 in.; crushes to corn size or dust.
10 × 7	five	1175	800	6,750	7	0	4	3	20	6	275	5	Takes pieces 7 in. thick; crushes to $\frac{3}{4}$ in.
15 × 5	six	1070	1200	8,000	7	0	4	8	24	8	275	8	Takes pieces 4 in. to 5 in. thick; crushes to $\frac{3}{4}$ in.
15 × 9	seven	2480	1400	12,860	9	0	5	0	24	10	275	9	Used for road ballast, for furnaces, breaking small for other crushers.
15 × 11	eight	2600	1400	13,000	9	0	5	0	24	10	250	9	Takes pieces 15 in. by 9 in.; crushes before smaller sizes.
15 × 13	eight	3000	1400	14,000	9	0	5	0	30	8	250	10	Takes pieces 15 in. by 9 in.; crushes before smaller sizes.
30 × 5	twelve	2600	1200	13,500	9	0	6	3	30	8	275	10	Takes pieces 20 in. by 10 in.; crushes to 1 in. or less.
20 × 10	twelve	3125	1500	15,000	9	0	5	6	30	8	250	10	Used for road ballast, for furnaces, and breaking small for other crushers.
20 × 15	fifteen	5550	1600	24,000	10	2	7	2	30	10	250	12	Used for crushing large masses to sizes suitable for blast furnaces, or before other crushers.

Dodge's crusher is often used in Colorado; it is not so well known as Blake's, but as it is very compact, and particularly adapted for taking quick samples, as it has both sieves and rolls attached, a description of it is given. For general purposes, however, it will not do as much work as Blake's crusher. It consists of two jaws *a*, Fig. 138, which are bolted to their respective positions, one being fixed and the other movable on a lever *c*, moved by an eccentric *h*. The box *d* in which the fulcrum *e* of the lever *c* moves, is adjustable horizontally by set screws *f*, and washers or plates *v*. The crusher proper is fastened to a stand to which the rollers *m* and *p* and the sieves *t* are attached. It is fastened by bolts *g*. A flywheel is attached to the shaft *i* upon which the eccentric *h*, the pinion *k*, the ratchet wheels *x*, and the flywheels *o*, with the tight and loose pulleys on the opposite side, are attached. The spurwheel *l* is driven by the pinion *k* which is keyed to the shaft *n* of the roller *p*, which causes it to revolve. The other roller *m* is moved by a gear on the opposite side of the crusher. Its shaft is in a box adjustable by a set screw

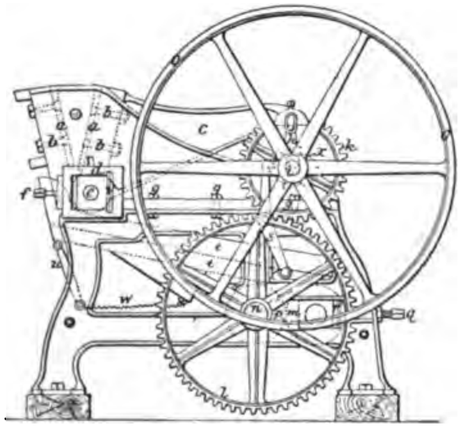


Fig. 139.

q by a rubber spring *r* backed with a cast-iron washer or cap between this box and the screw *q* so as to give elasticity to the roller. There are four sets of gear, for the purpose of using the rolls until they are completely worn down. The sieve *t* is moved by a ratchet lever *s*. The shaking table *t* containing the sieve, is for the purpose of taking out that part of the ore already

crushed between the jaws, and to relieve the rollers from the work of passing it through them again; also to distribute the ore evenly to the rollers from end to end, so that they should wear evenly. This shaking table is supported by a lever *u* under the jaw of the stamp. The motion of the screen is assisted by a spring *w*. The crusher is provided with patent soft wrought-iron crushing jaws *a*, which it is asserted wear longer than the hard cast-iron ones. The tyres of the rolls are made of steel, which does not wear into grooves and ridges as easily as cast iron. The Dodge crusher generally weighs 7500 lb., and requires six to eight-horse power. It crushes two tons of ore per hour. The flywheel makes 200 revolutions per minute. Below is a Table of the sizes of the Dodge crusher.

No.	Size of Jaw Openings.	Diam. of Pulleys	Width of Belt used.	Horse-power required.	No. of Tons per Hour, Nut Size.	Revolutions per Minute.	Weight complete.
	in.	in.	in.				
1	4 × 6	20	4	2 to 4	½ to 1	275	1,200
2	7 × 9	24	5	4 to 8	1 to 3	235	4,300
3	8 × 12	30	6	8 to 12	2 to 5	220	5,600
4	10 × 16	36	8	12 to 18	5 to 8	200	12,000

When the ore is screened, the screens should always be arranged to deliver the large pieces in front of the crusher, and to discharge the screenings into the stamp hopper. In such cases, one Blake crusher is more than sufficient to do the whole work of any of the sixty stamp mills of Nevada. It is even proposed with the perfected system of multiple jaw crushing to do all the fine crushing with a number of these machines arranged in a system.* When the ore comes from the mine in very large pieces, there should be two series of crushers of different sizes at different levels; in which case the large size crushers take all the ore which falls from their jaws directly into troughs leading to iron screens which allow the ore small enough to go to the stamp hoppers to pass, and discharge the rest into the jaws of the next size crusher, and so on. One large crusher will then feed into two small ones. This supposes the mill to be

* Trans. Am. Inst. Min. Eng., vol. xiii., p. 210.

built on the side of a hill and to have plenty of fall. The first crusher should break the ore into pieces of about 2 square inches; this will allow flat pieces of this size to pass, but they will be broken in the crusher below. All the ore should go through this breaker in wet crushing, and in order to keep down the dust, a small stream of water may be allowed to run through it. The object of this is to prevent injury to the machinery, and also to save the dust, which often contains the richest part of the ore, as has frequently been found by assay in some of the mills in Colorado. In the lower crusher the jaws should almost meet. This breaker should feed by shoots into a pocket which should connect with the battery. The dies of this breaker weigh about 250 lb., and last about three months when not broken by accident. By using two crushers the consumption of iron will be reduced to 0.3 lb. per ton of rock stamped. The great gain of using a crusher is shown by the experience of the Owyhee Mill, the stamps of which weighed 650 lb., dropped 60 times a minute, with an average fall of 8½ in.; the shoes and dies were used until they were worn so thin that the drop became 10 in. By breaking the ore by hand to medium size, the mill crushed 28 tons a day; breaking small by hand 30 tons; breaking large by Blake's crusher 33 tons, and breaking fine 37 tons. By increasing the rate of speed to 93 and 95 drops per minute, and using a coarse screen, the production was increased to 48 tons. The reason for the great increase is that the stamp is forced to do only its own work. All the ore that is already fine enough is sifted out after passing the jaws of rock crusher, so that any portions which will pass the stamp screens do not go into the mortar. By the introduction of properly constructed screens below the crushers, the product of the Tombstone Mill was increased 12 to 15 tons per day. In the Brunswick Mill each crusher requires one man and a helper to draw the ore out of the shoots. At the Consolidated Virginia the arrangement of the mill is such that one man does all this work.

The floors of the mill should be kept clean, and should be frequently washed, the result of the washing being caught in the agitators. No loose mercury should be found on the floors, tables, or anywhere else. The mercury should properly never

be touched, but in any case the whole floor of the mill should be washed at least once a day. All the drainage of the floors should be towards the agitators, and all drains should empty there.

The stamps always occupy the long end of the building, and are arranged in batteries of five stamps each, four batteries being usually placed together with a passage way between them. Occasionally only two batteries are placed together. In some mills both arrangements are adopted. The Brunswick Mill has 56 stamps; four sets at each end, or eight altogether, are arranged with two batteries of five stamps, together making 40 stamps. In the middle there is one set of four, with four stamps to each battery, making 16. The Consolidated Virginia has three sets of four, placed together. The usual arrangement is, however, to make the series uniform both in numbers of mortars placed together, and of stamps in each battery. Opinions and practice differ with regard to the weight and speed of stamps, some favouring light weight and great speed, and others the contrary; but high speed requires that extra care should be taken with the bolts, and that everything should be kept perfectly tight, or the battery would soon rattle itself to pieces. The general experience is in favour of heavy stamps with great speed and short drops. By increasing the weight of the stamps at the Tombstone Mill, with a few improvements to the mortars and screens in addition, the product of the mill in 1882 was raised from 60 to 115 tons per day. It will generally be found that low feeding is best, with iron almost touching iron, but this requires a very skilful workman who will feed uniformly by sound. Under this system more stems will be broken than by high feeding, but even if 15 per cent. of the stems are broken every month, this will be compensated for by the increased duty of the stamps. In any case the ore ought to be delivered to the stamps of uniform size, so small, if possible, that one blow of the stamp head will crush it. If the ore is large it may require several blows. The larger the pieces in the mortar the less the drop will be, and consequently the less effective the work of the stamp.

In the battery only just enough water should be used to keep the screens clear. This will vary with the ore; the more it is likely to pack, the more water must be used. When

water is scarce low feeding is necessary, as the mortar would otherwise soon become choked. If too much water is used the quantity of slimes will be largely increased. There should be as much fall as possible from the battery to the tanks, so that there will be no danger of the launders becoming choked. The amount of water required is usually 250 to 300 cubic feet per ton of rock treated, or about 10 lb. per minute. In the Brunswick Mill the supply is given by a 3-in. pipe. The Eureka Mill uses 12 miner's inches. This includes all the water used in the mill. It is supplied in Nevada by the water companies, and is measured by the miner's inch.

As it is impossible to have the feed perfectly constant when regulated by hand, automatic feeders* are being introduced. This is the most economical method where the ore is broken uniformly. As their work is constant they increase the output of the stamp, and at the same time decrease the wear. The jar and noise of the battery, and the fatigue of constant attention, is such that the men who feed, generally become careless. It requires a tough and intelligent man to stand the work for twelve hours, but if the work is pressed, and the mill large, no automatic feeder can take the place of a faithful and intelligent man, who will feed his battery so low that the shoes and dies almost meet. In the Brunswick Mill, where there are 56 stamps, and no automatic feeders, three feeders on shifts of eight hours are required to do the work of the mill.

The condition of the ore as it leaves the stamps varies according to the size of the screens. A mechanical analysis of the ore of the South Aurora Mill,† Nevada, consisting mostly of chloride of silver, and yielding $78 to $128 per ton, is given below.

Water	0.400
Metallic iron	0.006
Silver	0.145
Remained on No. 40 screen	0.025
" 60 "	1.978
" 100 "	16.150
Passed 100 "	81.296
	100.000

* See page 181.

† Mining Commissioners' Report, 1872, page 192.

This ore has the following chemical composition:

Silica	49.600
Carbonate of lime	48.808
Sesquioxide of iron	0.600
Alumina	0.400
Magnesia	trace
Chloride of silver	0.192
Water	0.400
	100.000

There is a great variation of practice with regard to battery amalgamation. It is thought to be necessary by some, in order to catch most of the native metal in the mortar, leaving the sulphides and chlorides of silver to be acted on in the pans. Many of the ores of Nevada contain some free gold, most of which is caught in the mortar if amalgamation is practised there. This is a constant temptation to the workmen to steal, as it is much more valuable than the silver amalgam, and is much more easily carried off in small quantities. Its only real advantage is in the possibility of catching some of the float gold which might otherwise be lost in the battery tailings. The loss where there is no battery amalgamation, is found to be about one per cent., most of which will be regained in the slimes. In the mortar the mercury is floured, besides every casting is full of holes and flaws, which become filled with a hard dry silver or gold amalgam, and a considerable amount is often collected from the battery pieces, amounting in some cases to several ounces; and even the most careful workman cannot prevent a certain loss, either because it cannot be picked out, or because it escapes observation. Every crack or notch in the wood or ironwork about the mill becomes filled, and it is very doubtful, with poor ores, if more is not lost than saved. If the ore is properly prepared the pans will catch almost everything.

The discharge from the battery will vary according as the mill is for wet or dry crushing. In dry crushing the whole front of the mortar is closed. The ore from the screens is discharged upon a horizontal endless table, which carries it to a bin from which it is generally elevated by an endless chain with buckets, to a hopper which discharges into the furnace where it is to be worked. In wet crushing the splash-box ends in a sluice, which carries the pulp into vats, where it collects. When the ore re-

quires roasting it is always first roasted and then chlorurised, the object of which is to convert all the silver into chloride, which is easily decomposed by the reactions of the pan, and readily yields the metal to the mercury. There have been a great variety of furnaces invented to roast the ore. The difficulty in doing it is the expense of labour. A great many attempts have been made to do away with hand roasting, and hence the invention of Brückner's cylinders and other mechanical furnaces, and of the Stetefeldt furnace.* Experience has shown, however, that when the ores are very "heavy," that is, contain a very large amount of sulphur, mechanical roasting furnaces cannot usually compete with the old-fashioned reverberatory furnaces, with long hearths. When the ores are light, that is, contain but little sulphur, they all work well, but the Stetefeldt furnace, which works admirably in Nevada on light ores, could at one time only chlorurise up to 50 or 60 per cent. in Colorado. At the Stewart Mill, where the ore contained 40 per cent. of sulphur, the tailings averaged 35 oz. of silver, which did not pay to work, but they were by far too rich for the economical reduction of the ore, and on this account the Stetefeldt furnace was torn down and reverberatory furnaces substituted for it, which burned five cords of wood for 12 tons of ore in twenty-four hours. Working three shifts three men are required. Since the addition of the auxiliary fireplace in the flue of the Stetefeldt furnace, such working would not be possible.

When the ore is roasted a chloruration assay is made of every charge of roasted ore. A shovelful, or 2 lb. or 3 lb., is taken, from which one-tenth is used for the assay. This is leached on a filter with hyposulphite of soda, until all the silver that is in a soluble condition is dissolved, which requires from twelve hours to twenty-four hours. The filter is burned and calcined with its residue, and an ordinary assay made of it. At the same time, an assay of the unleached ore is made, making two assays, one on the leached ore and residue, and the other on the ore as it comes from the furnace. The whole amount of silver in the ore is thus determined. The amount of silver that can be leached out shows the character of the roasting.

From the front of the cooling floor in dry crushing, and from

* Chapter V.

the front of the stamp down, in wet crushing the floor of the mill should be made double, to prevent any loss of ore, and especially of mercury or amalgam. In wet crushing mills the floors are tarred so as to be water-tight. In the best of the recently constructed mills, the floor of the stamp level is made to incline to a central trough under the battery floor, which connects with the pulp tanks. It is washed down two or three times a day. In the pan room the floor between the pans and settlers is made to incline towards the centre, and communicates with a settling vat, so as to catch any mercury or amalgam. This is swept up several times a day, whenever there is any accumulation. This is a necessary precaution against the loss of mercury and amalgam. The pan-room with its settling vats is arranged, in wet crushing, either at right angles to the stamps, as in the Consolidated Virginia, Figs. 133 to 137, and Eureka Mills, or parallel to them as in the Nederland Mill, Figs. 128 and 129, the Brunswick Mill, as shown in Figs. 130 and 131, and the Lexington Mill, Fig. 132.

In wet crushing, the quantity of water used is from $\frac{1}{4}$ to $\frac{3}{4}$ of a cubic foot per minute, the smaller amount being used for quartzose ores. The ore from the mortar is discharged into a splash-box which connects with a trough that inclines in such a way as to deliver the pulp most effectually. It is usually arranged in two sets, each one of which inclines from the outside towards the centre of the mill, and delivers the pulp into the settling vats. This main sluice delivers into another running along the settling vats with a discharge for each vat, which is closed by a wooden gate when it is full. These sluices are 6 in. to 8 in. deep. Their width depends upon the number of mortars discharging into them. In the Eureka Mill the discharge is from the battery into a sluice 10 ft. long, 6 in. to 8 in. deep, with an incline to the centre. Here it enters a sluice running at right angles to the mill, made into compartments 6 in. by 6 in., and arranged with gates, so that the stream can be made to diverge into any vat by means of an inclined trough regulated by a gate. The fall of the main sluice is 6 in. in 12 ft. In the Brunswick Mill there are 17 pulp vats which communicate with 13 slime vats, 8 ft. long and 4 ft. wide. In the Consolidated Virginia there are 42 pulp and 42 slime vats, 9 ft. long, 5 ft. wide, and 3 ft. deep, each

pulp vat having its own slime vat. In this mill the pulp and slime vats are on the same level, and for this reason, as they do not have to be transported from a lower level, the slimes are treated with the pulp, and not as formerly kept separate. In the Eureka there are 16 pulp vats, eight on each side of the floor, in double rows. They are 7 ft. by 10 ft., and 3½ ft. deep. From the slime vats in the latter mill, the overflow either goes to the slime reservoir, where it is again settled, or goes to waste. This reservoir is 500 ft. long and 60 ft. broad. There will not be less than 5 to 10 per cent. of battery slimes from the Comstock ores. There are generally 50 to 100 tons of pulp at a time on the settling floor, and often more than this.

The arrangement of these vats is, as we have seen, either at right angles to the stamps or parallel to them. The way they are to be arranged is determined by the lay of the ground, or by the same causes which influence the arrangement of the pans. There should be as many as possible, in order to do as much of the settling under cover as can be done, so that there shall be no stoppage on account of the weather in winter, or careless handling on account of the cold. They are always arranged so that as soon as one vat is emptied, the overflow from the others can be turned into it. When the mill is working with its proper complement of men, not more than three or four vats will ever be full of sand.

The work of discharging goes on constantly. The pulp is lifted out with a light iron bucket 1 ft. in diameter, with a gas-pipe handle about 8 ft. long, and thrown upon the floor behind the pans. They are of the consistency of very thick mud, and are arranged in such a way as to drain a little, but this is not necessary, as they are wet again when they go back into the pans.

The greatest possible number of the vats should be kept settling, but as every tank, before it is discharged, must be settled, at least two and generally three or four are full at once. This settling is necessary, so that the pulp will be sufficiently consistent to be thrown out on the drying floor. This is done when the vat is full. The pulp is then turned off and the vat settled, and the water baled or pumped out. As soon as the

pulp is consistent enough to be handled, it is thrown out on the drying floor behind the pans, which drains into the slime vats. This floor is 18 in. deep, and 8 ft. to 10 ft. wide, the length depending on the length of the building. At the Eureka Mill it is 65 ft. long. At the Consolidated Virginia it is 5 ft. 6 in. wide, and the length of the mill; it projects there 3 ft. over the slime vats. At the Eureka Mill, five men are required to handle the pulp. Such a large proportion of the work about the mill consists in handling the pulp, that it is now proposed to collect them in hopper-shaped vats above the level of the pans and discharge them by a slide into the pans.* It is also proposed to have the pulp overflow from one pan to another, and in this way avoid handling it.

The ore having now undergone all the mechanical treatment necessary, is ready for the pans. These are of various kinds, though the principle on which they are constructed is the same in all of them. Their capacity varies from 1500 lb. to 4000 lb. It may be said of them, as of all the machines, that the simplest in construction are generally the best. They are placed in front of the settling tanks in the wet crushing mills, and at a convenient distance from the roasting furnaces in the dry crushing mills. In the wet crushing mills, they are usually placed about 20 in. above the settling floor, and sometimes, as in the Brunswick Mill, project about 1 ft. over it. This projection is, however, not always convenient, as it contracts the space. The object of it is to prevent any loss of the ore between the edges of the floor and the sides of the pan. In some of the mills, when the pans have wooden sides, they are covered on the outside with sheet iron, which fits the woodwork at the bottom very closely. This is done to prevent the loss of mercury, from its forcing itself through the pores of the wood, or from leaks, so that all the mercury which comes through the wood, will be caught in the depression on the circumference of the bottom of the pan, between the wood and sheet iron. The usual life of a pan is from five to six years.

According to the character of the ore the amalgamation consists of two different periods; the first one of which is grinding,

* Trans. Am. Min. Inst. Eng., vol. xi., p. 321.

and the second is amalgamation properly speaking. The object of grinding is to reduce the ore to an impalpable fineness, and to clean and brighten any native metals, so that they can be readily acted upon. The motion of the mullers mixes all the materials thoroughly together, so that the reactions of the chemicals which are added to the pulp may reduce any ore in chemical combination, and keep the mercury clean and bright, and prevent it as much as possible from flouring. The object of amalgamation is to catch all the precious metals in the mercury. The pans are all intended to receive only well-ground ore, though in certain cases pieces as large as a pea may be treated, but this is a very questionable practice; it is better to have all the pulp fine.

The objects sought for in all these different machines, are to unite as far as possible a grinding with a stirring surface, and to combine at the same time a uniform wear of the cast iron, with such a distribution of the mercury, that it will not flour, and also that the pan shall have the greatest possible grinding surface consistent with cheapness and simplicity, and allow of the maximum of freedom in the circulation of the pulp in the machine. A few pans have been invented to treat large-sized ore, but these have generally proved failures, for the conditions favourable to reducing the ore from the size of a grain of corn to pulp, are such as are favourable to a maximum loss in mercury. Long experience has shown that the attempt to make the pan grind is a mistake, and that the best results are always obtained when the stamp crushes so fine that in the pan the shoes and dies can almost touch. In some mills, where no grinding is done, the iron shoes, when they wear out, are replaced by wooden ones. Simplicity and cheapness is claimed for all these pans, and they each propose to work the greatest amount of pulp, and produce the best results in a given time, with the least labour and cost of repair. All these desiderata are, however, not found in perfection in any one pan. Formerly pans with conical bottoms were very much used, but latterly they are being abandoned for flat-bottomed ones, because they are better adapted for grinding. With the maximum amount of work the flat-bottomed pan will wear more uniformly than the conical one, and is more easily put in repair. The conical pan requires less power, but this is

only an apparent advantage, as it does less work. The sides of the pans are generally made of cast iron, cast in one piece with the bottom. Sometimes the pans are made with cast-iron bottoms and wooden sides, in which case there is always a ring cast on the bottom of the pan for the staves to fit in. In very early times they were sometimes made with wooden sides and stone bottoms in imitation of the old Mexican arrastra, but this construction has been abandoned. The main object of the pan is to create such a current in the pulp and quicksilver that every particle of ore shall at the same time be brought into contact with every particle of quicksilver. This is effected by the shape of the shoes and the openings in the muller, which produce currents which must be strong enough at the bottom to move the quicksilver, while it throws the pulp high up on the sides of the pan, leaving the moving mass concave in the centre. The pulp slides down in the centre, passes under the muller and is thrown up again. If the current is sufficiently strong to carry the mercury, its rate of flow will not be the same as that of the ore, and it will tend to lag. If the velocity is properly proportioned, the mercury will break up and mix itself with the ore. To prevent the pulp rising too high on the side, wings of the approximative shape of an inverted ploughshare are attached to the sides of the pan. The upward current is thus checked and turned back to fall again under the muller. Each variety of pan claims to bring about all of these movements in the greatest perfection and thus to attain the best results. With regard to the mechanical part of the pan, the driving wheels below should be as plain as possible and open to inspection. The muller must not be too much cut away for fear of weakening it. The raising and lowering of the muller when done by a large handwheel should have a left-hand screw on the top. The jam wheel should be of the same size as the raising wheel instead of being smaller, as it usually is. It often requires more power to lower the muller than to raise it.

The pan consists of several distinct parts; the pan proper, upon the bottom of which the dies are fastened, and the muller, upon the bottom of which the shoes are fastened. In their general detail the pans most in use are alike, differing chiefly in

the arrangements and number of the shoes and dies, and the method of fastening them either to the muller or to the pan bottom. The muller itself will vary in shape and in its mode of suspension, and this is a matter of some importance, since upon the mode of suspension will depend the ease with which the muller may be raised and lowered to effect or prevent grinding or to remove it entirely from the pan, in order either to clean it or to replace the shoes and dies. In its simplest expression the muller is a flat disc attached to a cone whose surface is more or less pierced in the different varieties. In general, the nearer solid it is the stronger it will be. When it is too much cut out it is liable to break. The number of shoes to be attached varies from four to twelve. They are from 2 in. to 3 in. thick, and weigh 500 lb. to 800 lb. Experience has shown that it is desirable to have at least six. With a less number the movement of the pulp is not favourable to the highest amalgamation, and with the higher the muller is liable to become weak and to break. Some of the pans have a double bottom, and more rarely jacketted sides, for the introduction of steam, in order to raise the temperature of the pan to 200 deg. Fahr., to assist the amalgamation. This complicates the construction of the pan, and increases its cost. Others do not have any false bottoms, as at the Brunswick Mill, but introduce the steam directly into the pulp, by a pipe which goes to within 5 in. or 6 in. of the bottom of the pan. This was formerly thought a disadvantage, as the steam tends to dilute the pulp and make it too thin. It has, however, the great advantage of saving time, as the charge is raised much more rapidly to the proper temperature in this way than by the more complicated system of a double bottom, whose only advantage is the saving of fuel effected by the use of the exhaust steam. Direct steam must be used in the pan, as exhaust steam always carries oil with it and would thus prevent the action of the mercury. The pans are generally covered with wooden or cast-iron covers, and the steam introduced through a hole in the cover. At the Brunswick Mill there are for this purpose two boilers, only one of which is worked at a time, which give steam at 25 lb. pressure, but 80 lb. is sometimes used here and at the Eureka Mill. This amount of steam power is necessary, as in

winter not only the pulp in the pans but the water and the whole mill must be heated. The wood for this purpose costs $5 a cord. The temperature of the pulp in the pan is brought up about to the boiling point by the steam. Up to this point the hotter the pulp the more perfectly the reactions will take place. If the pans are covered, as they always should be when steam is used, the pulp will be kept at this temperature, after the steam is cut off, by the reactions which take place during the operations in the pan. Sometimes, especially in such mills as have had pans with

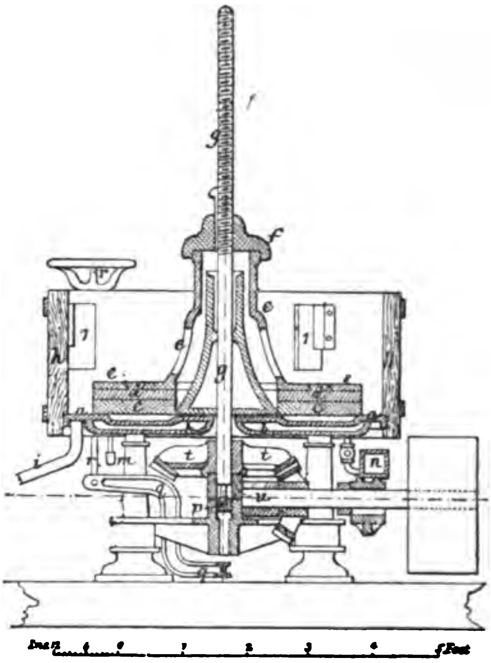


Fig. 140.

double bottoms constructed, both methods are used, and the steam is introduced into the pan and underneath it at the same time. It has been considered an advantage in some cases to use the double bottom, because in that case the exhaust steam may be used. Where the steam is used directly, it cannot pass through the engine on account of the oil and grease which exhaust steam contains, which would hinder the amalgamation, but must be taken directly from the boiler. In some mills amalgamated copper plates are used in the pan and much amalgam collects

on them, but the general rule is to discharge the pans into settlers and collect the mercury and amalgam there.

There are a great number of pans in use, which differ more in detail than they do in principle. Almost all the pans with conical bottoms have been given up. Only those are described which are in general use either in Colorado, Nevada, or California. They resemble each other in many particulars, differing in mere detail of construction, or of the arrangement of the muller, shoes, and dies. Most of the early pans have, as stated above, double bottoms for steam, which are gradually being given up for the introduction of steam directly into the pulp. A 5-ft. pan with wooden staves will weigh about 6500 lb.; an iron pan of the same dimensions about 7000 lb. When required for use in mountainous countries they are made in sections, the heaviest piece of which does not weigh over 300 lb.

The Wheeler pan, Fig. 140-141, is one of the oldest, having been

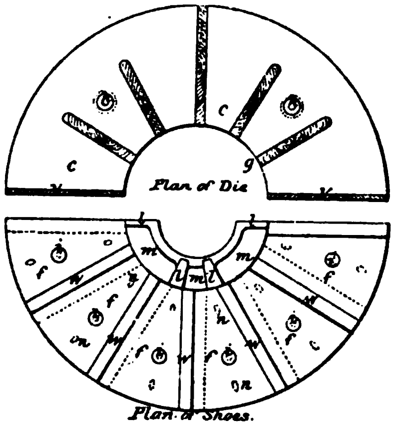


Fig. 141.

introduced in 1862. It is usually made 5 ft. in diameter, with a flat iron bottom, and a steam chamber *b*. The sides are of wood, or of cast or wrought iron. When they are of wood, the bottom is let into the sides. The dies *c* are fastened to the bottom by dovetails which fit into sockets. The muller *e* is carried by a vertical shaft *g*, which passes through a cone in the centre of the pan; on the upper end of which a screw thread is cut for the purpose of raising the muller, when the pan is to be cleaned. It is secured at any height by a key fitting into a slot

in the muller nut *f*. The shoes *d* are attached to the muller by dovetails and sockets, and the distance between the shoes and dies is regulated by the handwheel *r*, which raises or lowers the block *p*, on which the axis of the pan rests. On the sides amal-

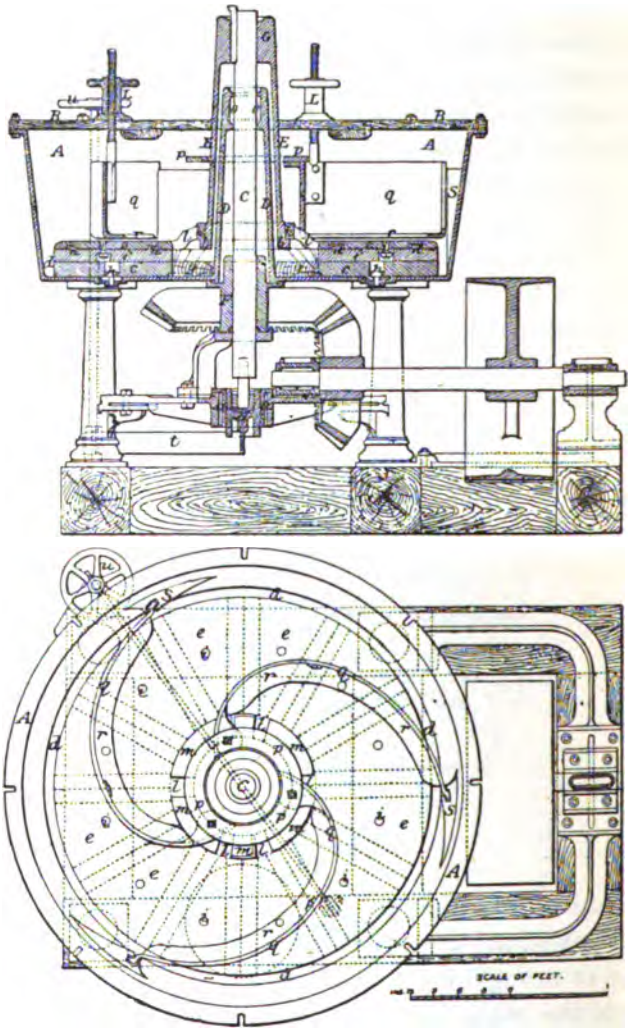


Fig. 142.

gamated copper flies or wings are placed, whose object is to constantly change the motion of the pulp, and to catch any amalgam. The motion is communicated by a bevel gearing. The pan can be stopped or set in motion by means of a handwheel attached to the

lever *g*. The peculiarities of this pan are, the means of raising the muller, and the great distance between the muller and the sides of the pan.

The Varney pan, Fig. 142, was at one time the most extensively used of all the pans, but is now going out of favour. It is made of iron, the sides being either cast in one piece, or made sectional. It has the same suspension as the Horn and Patton pans, but the muller is made in one piece, the dies are bolted to the bottom of the pan, and the shoes are also bolted to the muller. It is consequently not so easy to remove them as in the pans where they are fitted with slots, like the Wheeler pan. Only a part of the bottom of the pan is covered by the die. It has the same wings as the Wheeler pan.

The Horn pan, Fig. 143, has iron sides and a flat bottom like the

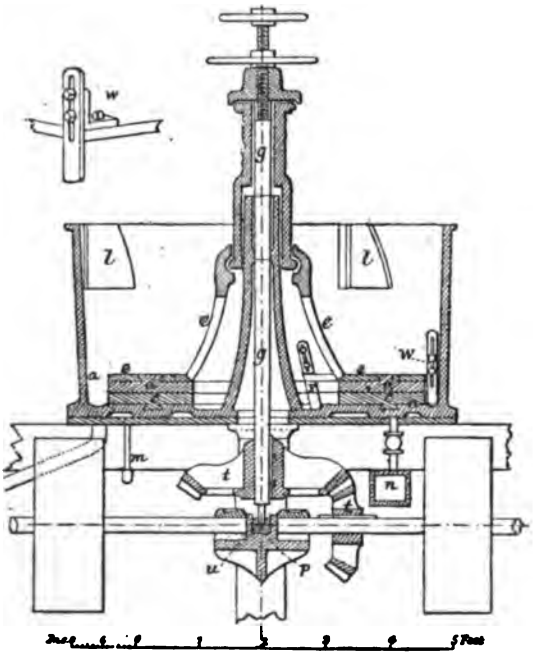


Fig. 143.

Wheeler, and also has a steam bottom, which is composed of a flat plate *b*, upon which the pan proper is bolted. The space for the steam is made by the two hollows formed by the projections necessary for the sockets of the die. The shoes and dies are

fastened in the same way as in the Wheeler pan. The muller is not, however, fastened to the driver, but is caught in grooves, so that it moves only when the driver is turned in one direction; in the other it is loose. The height of the shoes and dies is regulated by two handwheels at the top of the shaft. On the circumference of the inside of the pan, there is a groove into which a scraper *w* is introduced; it is fastened to the muller as shown in the separate sketch. A scraper is also attached to the inside of the muller, fitting into a similar groove there. The shape of the wings is different from that of the Wheeler pan. A yoke is attached to the bottom of the pan which carries the bearings for the vertical and horizontal shafts. This yoke and the scrapers are the peculiarities of this arrangement.

The Patton pan, Fig. 144, is a combination of the Wheeler and

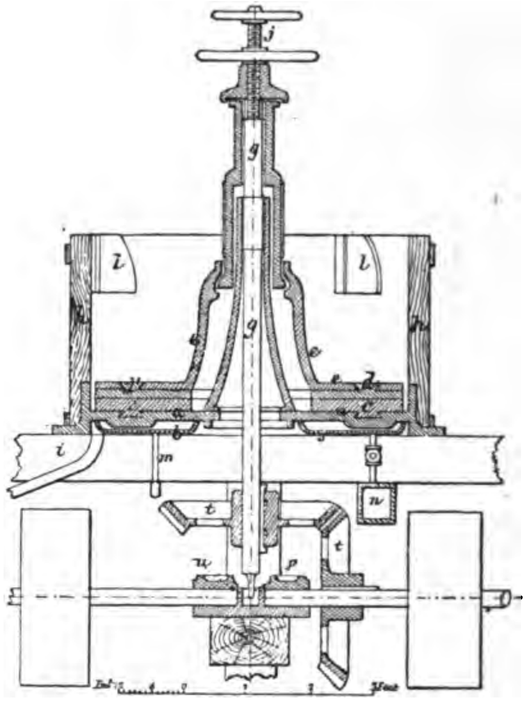


Fig. 144.

Horn pans. The steam bottom is fastened in the same way, but is not exactly like that of the Horn pan. The sides are of wood supported by a flange, which is nearly as high as the muller,

in order to prevent the loss of mercury, and is a much better disposition than that adopted in the Wheeler pan. The shoes and dies are attached as in the other pans. The motion of the muller is the same as in the Horn pan, as well as the means of regulating it. It has the same wings as in the Horn pan, but no scrapers. The bearings for the gear wheels and horizontal shaft are set on a beam underneath.

The Combination pan, Fig. 145, is a combination of the Wheeler

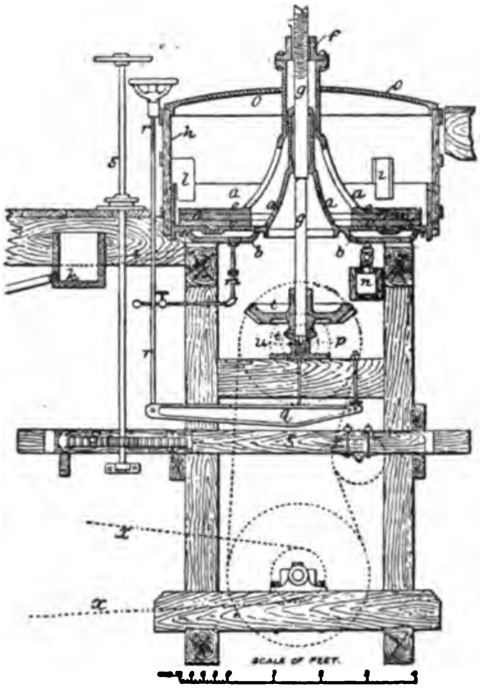


Fig. 145.

and Patton pans, with a number of improvements on both. Its capacity is 15 tons in twenty-four hours; it is the most extensively used of all the pans. When free milling ores are treated, the sides and bottom are cast in one piece; when used for roasted ores the sides of the pan are of wood. From the top of the iron rim about 6 in. of the sides is lined with $\frac{1}{8}$-in. copper plates, and the wings have also copper plates, 16 in. wide at the bottom and 10 in. at the top bolted to them. This is done to help the collection of the amalgam, which is always finer on the copper than on

the iron. Some of the chief advantages of this pan are that the muller is loose from the muller stem, and that both the muller

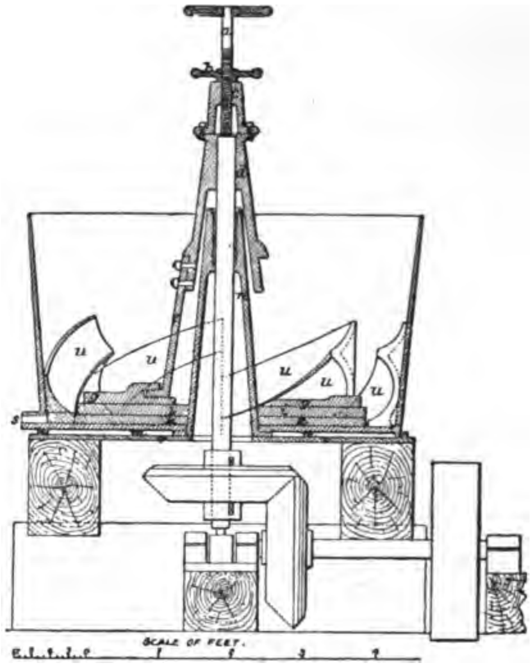


Fig. 146.

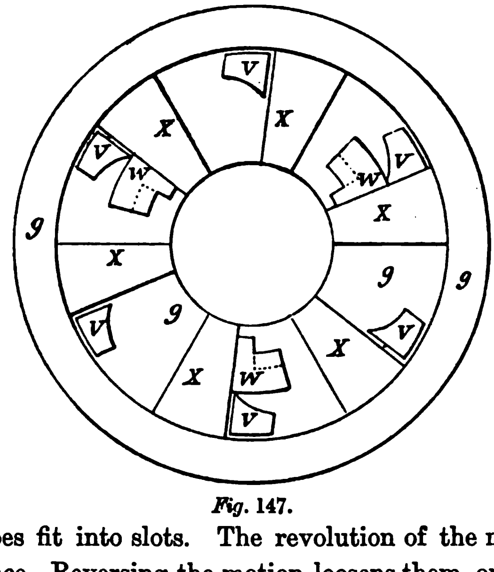


Fig. 147.

and the shoes fit into slots. The revolution of the muller keeps them in place. Reversing the motion loosens them, or they can be

easily pried out with a bar when necessary. The muller is fixed in the stem by means of a key in the screw thread; by taking out the key and turning the muller on the stem it may be raised and kept at any height for any length of time, and as easily lowered. The three wings are attached to pieces of iron, which are dovetailed to fit into slots in the side of the pan.

Stevenson's pan, Figs. 146 to 149, is one of the most recently introduced, and is said to be the best pan for amalgamating that has been invented. It does not have some of the mechanical advantages of the Combination pan, and has less grinding surface,

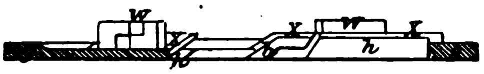


Fig. 148.

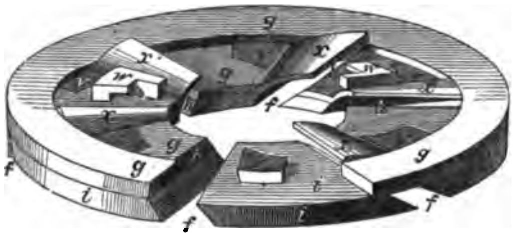
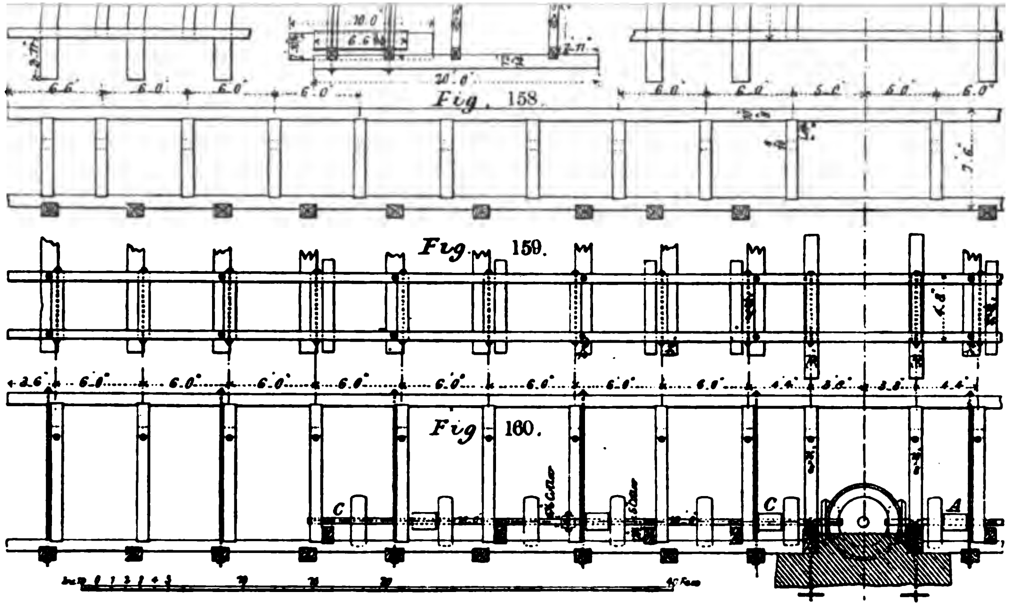


Fig. 149.

though the distribution of the pulp throughout the pan is much more perfect than any other, on account of the four double curved mould boards *u*, which are introduced to throw the pulp to the outside and upper part of the pan, from which it falls again under the muller, and so insures a maximum amount of grinding. As the mullers are nearer the centre of the pan, less power is required to do the grinding than in other pans of the same capacity. The mould boards raise the pulp regularly without violence, and admit of a larger charge in the pan. It has a steam bottom like the other pans, and the suspension is the same as in the Horn and Patton pans. The muller *g* is attached to the driver *d*, which is keyed upon the shaft by four lugs *e*, so that the space between them and the cone of the pan is entirely free. It has six shoes *i*, which weigh 100 lb. each, and eight dies *k*, which weigh 85 lb. each. The arrangements for driving the pan are the same as in the others. It has a capacity of 3500 lb. of pulp. Its total weight is 6500 lb.

Above the pans, as at the Brunswick, Consolidated Virginia, and all recently constructed mills, a track with a 12-in. gauge is attached to the framework of the roof, which carries iron rails. The cars running on this track have eye-bolts for Weston chain blocks, which are used to lift off the covers when necessary, and to take out the muller when the pan is to be cleaned. This arrangement is made not only on account of convenience, but also to guard against accidents, which were always happening to the feet of the men when an ordinary block and tackle were used. The muller or cover, or both of them, may be removed by the overhead tackle, carried to any part of the mill, placed on the floor or kept suspended, according to the requirements of the work. It is no trouble to run the car to any spot, while it was not easy to attach the block and tackle, so that the pans are more carefully watched than formerly. The arrangement of the pans in the Consolidated Virginia Mill is shown in Figs. 133 to 137, and 167 to 172. That of the Nederland Mill in Figs. 128 and 129, and that of a dry crushing mill built in 1873, in Figs. 130 and 131. The plan and sections of the Lexington Mill are shown in Fig. 132. The building of the Consolidated Virginia Mill which contains the pulp and slime vats is shown in Figs. 150 to 153, and the pan and settler frames in Figs. 154 to 160.

Before charging, water sufficient to cover the muller is introduced into the pan. If the pan is open, it is introduced from a discharge pipe above it. If it is covered, an india-rubber pipe is attached to the discharge pipe, which enters the pan, in the Brunswick Mill, through an opening in the cover 11 in. by 8 in. The charge of ore is then put in. A charge varies usually from 800 lb. to 4500 lb., the latter charge being usually for treating slimes in very large pans. At the California Mill, Virginia City, with the Combination pan the charge is 3600 lb. At the Eureka Mill, Nevada, where the Wheeler pan is used, the charge is 3200 lb. At the Brunswick Mill, near Carson City, where the Horn pan is used, it is 2600 lb. At Stewart's Mill, in Georgetown, Colorado, where the Varney pan is used, it is 1100 lb. In Judd and Crosby's Mill, Georgetown, where the Combination pan is used, it is 2800 lb. At the Nederland Mill, in Colorado, it is 850 lb. The charge of ore and water fills the pan about half



156. Plan of the settler frames.
157. End view of the pan and settler frames.
158. Side view of the settler frames.
159. Plan of the pan frame.
160. Side view of the pan frame.

To face page 380.]

full. The pulp becomes in this way about as thick as batter. In some mills, as at the Eureka, it is made as thick as the muller can move in.

In charging, it is usual to raise the muller about $\frac{3}{8}$ in. to $\frac{1}{2}$ in., in others it is not raised but bears on the dies, and the ore, roasted or not, is either dumped upon the floor in front of the pan and shovelled into it, which takes about five minutes; or, in the case of covered pans, the car bringing the ore contains the charge, and is arranged to dump its whole contents through an elliptical hole 18 in. by 20 in. in the cover of the pan, as in the Nederland Mills in Colorado, and the Brunswick Mill in Nevada. While the charge is being made the muller revolves from 60 to 90 revolutions a minute; at the Consolidated Virginia Mill it makes 90. At the Eureka Mill it makes 85, at the Brunswick 88, at Stewart's 70, at Judd and Crosby's Mill 60, and at the Nederland 75. It is absolutely necessary that the muller should be in motion when the charge is made, in order to save power, for the reason that if the charge was made directly upon the muller, it being motionless, it either could not be moved, or the force required to start it would be so great that it would be liable to break. The rapid revolution of the muller causes the pulp to rise almost to the top of the sides of the pan. As soon as the charge is made the muller is lowered so that the shoes and dies almost touch. Steam is now introduced to raise the temperature of the pulp from 160 deg. to 200 deg. Fahr., when it is shut off. The charge will be in the pan on an average about five hours. At the end of one to one and a half hours, the ore is reduced to a fine pulp. It is tested by feeling it between the thumb and fore-finger, and when the proper grain is recognised by the touch, the mercury is introduced.

To secure the best grinding the pulp must be thin; to secure the best amalgamation it should be about the consistency of honey, so that the best practice consists in having the pulp thin at first, but so arranged that when it is ground it will thicken sufficiently to cause the mercury, as soon as it is broken up into small globules by the motion of the muller, to become diffused and suspended in the pulp. In order to ascertain the exact condition of the pulp, it is usual to take an assay sample which should show the mercury

evenly distributed. If the pulp is of the proper consistency, all the small globules will be carried by the current which comes out from under the muller to the surface and returned again with the current, and the large globules will sink through the pulp and be thrown up again. The production of the most efficient current is the object and aim of all the inventions of pans, and hence the number of shapes of the shoes, the slots, openings, and grooves in and through them and the muller, to produce the continual discharge from under the shoes, thus throwing the pulp to the surface, and causing a return of the current under the shoes. In some cases the mercury is introduced from the commencement; this is a bad plan, for the reason that the mercury being ground with the ore is likely to become floured and lost.

The time of grinding is sometimes as long as four hours, when by accident rebellious ores which have been roasted with salt are not well chlorurised, as it is often cheaper to reduce the ore in the pan than to re-roast it. Sometimes the muller is raised when the mercury is introduced, and sometimes it is allowed to grind until within three quarters of an hour of the time of discharging the pan; but generally the mercury is not added until after the grinding is finished, which with free milling ores is two hours; with roasted ores it is generally added at once. The object of raising the muller when the mercury is introduced is to prevent its flouring. When the muller is raised after the mercury is added, the materials for quickening and saving it are added.

The quantity of mercury charged varies in different mills, and with different ores, according to their richness; it being generally about 60 lb. to 200 lb. to a charge of 1200 lb. to 1500 lb. of ore. In some cases as much as 300 lb. of mercury are put into the pan, and an additional quantity is put in towards the close of the operation. No very regular rule seems to be followed. The quantity should be increased or diminished according as the assay shows the working to be good or bad. At the California Mill, with the Combination pan, it is 350 lb. At the Brunswick Mill, with the Horn pan, the charge is 360 lb.; at the Eureka Mill, with the Wheeler pan, the charge is 200 lb. for ore assaying $75, and 160 lb. for that assaying $50, three quarters

being gold and one quarter silver. At Stewart's Mill, with the Varney pan, 1¼ oz. of mercury are added for each ounce of silver. At Judd and Crosby's Mill, with the Combination pan, 250 lb. is used for 100 oz. ore. At the Nederland Mill, where the combination pan is used, 150 lb. is charged.

There are several ways of introducing the mercury. Sometimes the flask is simply emptied over the side of the pan, which is a very bad method. Sometimes the mercury is poured from the flask between the fingers; sometimes it is poured through a strainer. In some mills it is pressed through canvas. The object of all these different methods is to scatter the mercury as far as possible, so that it shall not collect in any one place. The usual charge of mercury is about 10 per cent. of the average charge of ore. A small quantity of sodium, or of sodium amalgam, is in some cases added to the mercury to make it "quick." No special advantage seems to result from it, so that this addition has been for the most part abandoned.

Generally, before the mercury is introduced, the muller is raised so that the shoes are ½ in. from the dies, in order to avoid the grinding action which would flour the mercury; but in some mills, as at the Brunswick, the muller is always down, and is never raised, except to clean out the pan. This is, however, an exceptional case, for with any ordinary pan all the grinding necessary has been performed, so that when it is advisable to keep the muller down a maximum length of time, it will be raised at least three-quarters of an hour before the pan is to be discharged, as all the work that it can do to advantage is at that time moving the mercury about, and incorporating it in the pulp.

The time devoted to amalgamating, will be influenced by the richness of the ore, and to some extent by the capacity of the mill. It is often advisable to sacrifice a very small percentage of the value of the ore for the sake of working a greater number of tons per day. To insure economical results, a charge should not be allowed to remain in the pan, subject to the action of the quicksilver and chemicals, less than four or five hours. The results of almost all the experiments made on this subject show that but little is gained by lengthening the time. In many of the mills the entire time of the ore in the pans is limited to from four

to six hours; in others it is as high as eight hours. The same ore treated side by side often does not show a gain of one per cent. per hour after four hours' treatment. The most important part of the process is to keep the mercury clean. Some ores dirty the mercury, and in such a case, as it cannot be kept clean, it should at least be cleaned after every charge, and the pan should be washed as free from mercury as possible, before adding a new charge of it. The iron of the shoes and dies of the pan is necessarily more or less abraded by grinding, and corroded by the ore and chemicals introduced; this grinding and corrosion will sometimes amount to about 10 lb. for one ton of ore.

A large number of chemicals are employed partly to produce chemical reactions, and partly to keep the mercury bright, and thus to assist the amalgamation. The idea of using them was taken from the Mexican patio process, which was at first used in Colorado, but was afterwards abandoned. These chemicals were, in the earlier stages of the process, known as "doctor's stuff," and consisted of sage tea, tobacco juice, urine, and almost everything which ignorance or charlatanism could devise, and which could have no possible influence on the charge, in order to give some stamp of originality to the process used by the individual who invented it. The chemicals which are now used are salt, lye, nitre, cyanide of potassium, lime, "blue stone," or sulphate of copper, sulphuric acid, and sodium amalgam. They are sometimes introduced with the ore, when it is very refractory, and are intended to produce chemical changes in it, and sometimes are added only with the mercury, as with ores which are easily treated, when their principal object is to keep the mercury clean and bright, and only incidentally to produce a decomposition in the ore. There does not seem to be any definite rule in this respect, for many mills with refractory ores do not charge the chemicals until after the mercury, and some with light ores charge them with the ore. The quantity used varies at different times and in different mills, from 1½ lb. to 3 lb. or more for each charge of ore according to the richness of the ore. At the California Mill 12 lb. to 24 lb. of salt and 4 lb. to 8 lb. of blue stone are used. At the Consolidated Virginia Mill 6 lb. of salt to

the ton is introduced with the charge, and twenty minutes afterwards 3 lb. of blue stone to the ton. At the Eureka Mill 1 lb. of sulphate of copper per ton of ore is introduced with the charge. At the Brunswick Mill a handful of salt and a dipperful of sulphate of copper is used. At Stewart's Mill 1½ pints of a solution of 32 lb. of sulphate of copper, 6 lb. of lye, and 4 lb. of nitre, dissolved in 12 gallons of water, is charged immediately after the mercury is put in. At Judd and Crosby's Mill 5 per cent. of salt for 60 oz. ore, and 10 per cent. for 200 oz. ore, is put in at the commencement. At the Nederland Mill one gallon of lime water is introduced into the pan before the charge. If it is found that there is too much, its effect is afterwards counteracted by a weak solution of sulphuric acid. When salt is added it is generally charged with the ore, the amount varying from 5 to 10 per cent. When gold ores containing tellurium are added to the silver ores, 14 oz. of nitrate of mercury are sometimes used for 100 lb. of ore. The average fineness of bullion produced under the most favourable circumstances, having been once established, the quantity of chemicals to be added will be determined by the increasing or diminishing of the fineness of the bullion. The use of "blue stone" may almost be said to be a fashion, and at times, establishments for manufacturing sulphate of copper have been able to make very large profits, while at others they made none at all, there seeming to be no fixed rule as to whether more or less sulphate of copper should or should not be used, except that more should be used with very base, and also with very high grade ores, than with ordinary ones. The free use of chemicals is a decided advantage. When used with discretion, in the treatment of the ore, slimes, and tailings, the yield counted on the ton of ore can sometimes be brought up to 96 and 98 per cent., when without them it would not be over 80 to 85 per cent., which is the pan yield of the Consolidated Virginia. The more rebellious the ore, the greater the need of them, but they should be used rationally, and their effect checked by frequent assays of the pulp to ascertain it.*

* In chapter v. of vol. iii. of the "Report of the United States Survey of the Fortieth Parallel," Mr. A. Hague has made an elaborate report of his investigations on the chemistry of the pan process, to which those interested in the subject would do well to refer.

It appears, as the result of experience, that salt and sulphate of copper are the most important chemicals to be used; in most mills they are the only ones used. Salt is generally added with the ore. Sulphate of copper is added when the pan is at its full heat, or about fifteen minutes after the charge is introduced. It is established that a high yield of ore cannot be had without the liberal use of chemicals, and that all except those intended to save mercury should be introduced at the commencement. The chief influence of salt is to form chloride of silver from the decomposition of the sulphides, and this is very desirable, as this salt is very easily decomposed by mercury. When sulphate of copper is present, it produces besides, chloride of copper, which the iron partly reduces to sub-chloride; both these chlorides act on the blende and galena, and prevent the lead and zinc from passing into the amalgam. The sulphate of copper is useful indirectly in the production of chlorides, and also directly to decompose the galena and blende which would soon make the mercury sluggish if their action was not counteracted. The lead is transformed into an insoluble sulphate, and the metallic copper, set free, amalgamates and aids the action of the mercury. The fine particles of iron removed from the pan by the process of grinding and brought into the pan from the stamps, assist in the decomposition of any undecomposed sulphide which may be there, and aid in the decomposition of the chlorides of mercury and silver. It thus not only saves mercury by the decomposition of the salts formed, but also keeps it bright. Sodium amalgam is used to prevent the flouring of the quicksilver, occasioned by the presence of binoxide of manganese in the ore. The loss of iron in the batteries varies from 2 lb. to 3 lb. per ton of ore; in the pan it varies from 7 lb. to 10 lb., so that the total amount of iron in the pan in a state to produce these reactions, will be 19 lb. to 23 lb. per ton of ore, the largest amount being with roasted ore. In order to save the action on the pans, from 10 lb. to 20 lb. of iron turnings are sometimes added to the charge; it is undoubtedly one of the most powerful agents in helping the action of the mercury.

The method of discharging the pan differs in different mills, but only with regard to the collection of the mercury. In some mills

in Colorado and elsewhere, five to ten minutes after the pulp is thinned, while the pan is full of water, before the discharge of any of the pulp, the mercury is drawn into an iron box, 12 in. square, or into an iron kettle, by means of an india-rubber tube permanently attached to an iron pipe cast on the bottom of the pan, and which during the working is tied to its side. The pulp is then discharged into the settler, clear water running into the pan during the discharge, which takes about ten minutes.

In this case there are always about 50 lb. to 60 lb. of mercury remaining in the pan, but as what remains is used in the next charge, no mercury is lost from this cause. Nine-tenths of the mercury leaving the pan is caught in the discharge box, the rest runs into the settler. In some cases the whole contents of the pan is discharged into the settler. Generally about a quarter of an hour before discharging, the speed of the muller is reduced to about forty revolutions. Water is introduced so as to fill the pan almost to the top. This is done to cool down the pulp, and at the same time thin it so that the mercury can settle, clean water running in as the pulp discharges into the settler. If there is a mercury discharge in the bottom of the pan, most of the mercury is found there. It takes about half an hour to discharge the pan in this way, and get it ready for a fresh charge. The amount of water used is about 16 lb. per minute. After a charge is drawn, the pan, more especially if the ore is likely to soil the mercury, should be well washed with clean water to clear it of any pulp, and also to get as much of the mercury as possible out of the pan. It cannot all be removed from a flat-bottom pan; there will still remain from 50 lb. to 60 lb. There will be some between the dies, and some amalgam sticking to the sides of the pan, to the shoes, and on the muller, but this is of little consequence, unless the ore tends to soil the mercury. A clean up of the pan takes place in some mills once in two to four weeks, in others, where the shoes and dies wear rapidly, only when they are to be replaced. In custom mills it is made after the working of every lot of ore. All parts of the pan bottom are then examined, and the shoes, dies, and muller scraped, to detach the adhering hard amalgam. In order to do this, the muller is loosened and raised, in the Wheeler and Combination pans, by the screw to the top of the shaft, and in the other kinds

of pans it is lifted out by the overhead pulley, and is either suspended or put on the floor of the pan room. The wings are then removed. If the shoes and dies are worn out they are taken off and examined, and they and the pan bottom are carefully scraped and washed. The shoes and dies are not removed for a clean up, but only when they are worn out. The time of doing this work depends on the ease with which the different parts of the pan can be removed. The method of fastening the shoes and dies is one of the chief characteristics of the different pans.

Experiments have shown* that the amalgamation goes on rapidly at first, and then slowly. In an hour after charging the mercury, 74.66 per cent. of the silver was already acted on. At the end of the second, 76.26 per cent.; at the end of the third, 77.74 per cent.; and at the end of the fourth hour, 81.04 per cent.; nothing was gained by prolonging the operation after this.

The amount of amalgam obtained from a pan will vary with the ore treated. In Colorado it ranges from 20 lb. to 75 lb. In Nevada it is often as low as 15 lb., and in many mills is not more than from 20 lb. to 30 lb., the amount obtained depending on the time between the clean ups. The pans never stop when there is ore to treat, except for repairs or a clean up. Each pan is arranged so that it can be stopped when necessary independently of the others, so that repairs or a clean up may be made, if desired, on one pan at a time. The false bottoms of the pans wear about a year. They are always made separate from the sides, which wear very little, and are bolted on, so that they may be changed when necessary.

In the Brunswick and Eureka Mills the shoes and dies wear only thirty days. At Stewart's Mill they are removed when they wear down to $\frac{3}{4}$ in. They have to be removed every three or four months. At the Nederland Mill they wear five months. The whole pan must be generally renewed once in three years. The amount of wear on it depends on the character of the ore treated.

In the Brunswick Mill the power is communicated to the pans by a belt 11 in. wide, running over a wooden wheel on the cam

* Trans. Am. Inst. Min. Eng., vol. xi., p. 104.

shaft, 6 in. in diameter. These belts are always under the pan floor so as to be out of the way.

At the Eureka Mill it takes six men per shift of twelve hours to work the twenty-six pans. At Stewart's Mill there are seven men in twenty-four hours for twelve pans; one head amalgamator, one carman, one labourer, and two men per shift at the pans in twenty-four hours. Occasionally when the work presses, the shifts are made eight instead of twelve hours. At the Brunswick Mill four men are required, two on a shift for the pans and settlers. In Judd and Crosby's Mill two men, one amalgamator, and one helper, per shift of twelve hours, do the work of the four pans and two settlers. In the arrangement of the mill, one settler is provided for two pans. At the California Mill, which has forty pans running on ore and four on tailings, there are in twenty-four hours, six amalgamators, twenty slime shovellers, two men repairing the pans, two men to work the amalgam.

Generally two pans discharge together into one settler, but sometimes only one at a time is discharged, which is not a good plan, as it reduces the time of settling one half. The discharge sluice from the pan runs under the floor of the pan room into the settler. At Stewart's Mill this sluice is riffled, and some mercury is caught in these riffles.

The Table below gives the number of pans, &c., used in the different mills.

—	Number of Pans.	Kind of Pan.	Number of Revolutions of Muller.	Charge.	Number of Settlers.
				lb.	
Stewart	12	Varney	70	1100	6
Judd and Crosby	4	Combination	60	2800	2
Nederland	14	,,	75	850	7
Brunswick	26	Horn	88	2600	13
Eureka	27	Wheeler	85	3200	13
Consolidated Virginia	28	Combination	85	3500	14
California	40	,,	90	3600	20
Ontario	24	,,	65	2800	12
Lexington	20	,,	65	2800	10

The number of pans required in wet crushing is generally two to three for every battery of five stamps; in dry crushing two are sufficient, except in the case of very small pans.

The settler, Fig. 161, is sometimes only a wooden tub 8 ft. to 9 ft. in diameter. Generally it has revolving arms, a cast-iron bottom, and wooden or sheet-iron sides. The wheel which raises and lowers the muller should be right-handed, as that is much more convenient in case a belt slips, but when the muller drags, the power used on the wheel helps to move it. It is 9 ft. in diameter at the Eureka Mill, 10 ft. at the Brunswick, and 3 ft. deep, having either a flat, or, more frequently, a more or less

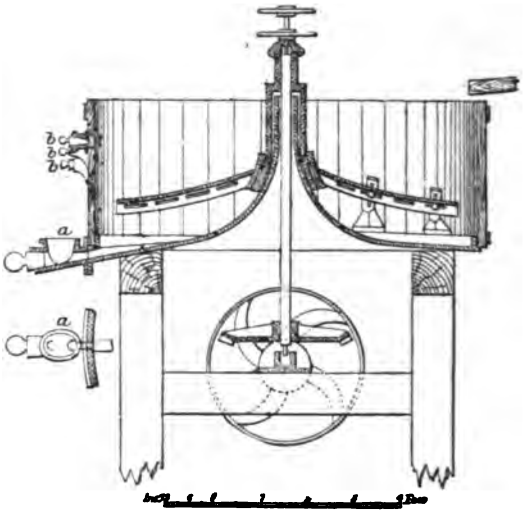


Fig. 161.

conical bottom with agitators. At the Manhattan Mill, where there is a settler for each pan, they are only 3 ft. in diameter. Its construction is similar to that of the pans, except that it is larger and has no dies. It has four arms or sweeps bolted on to the driving cone, to each of which two adjustable wooden shoes, shaped like cultivator shares, are attached, which reach to within $\frac{3}{4}$ in. of the bottom, and plough up the sand as it settles. The shoes last from one to six months. They are always kept at the same height, except when a pan is broken, when its contents are discharged into the settler, the shoes lowered, and the charge finished there. The charge from the pan fills the settler

about one quarter full. In the bottom there is a groove which commences at the centre, and is 3 in. wide and 2 in. deep at the circumference. This connects by a 1½-in. pipe with a bowl *a*, Fig. 161, bolted on the bottom of the side to receive mercury and amalgam forced into it by the pressure of the water.

It has three discharge holes *b* at different heights. They are placed inclined on the sides and not directly one above the other. These holes are closed with wooden plugs. The bottom of the settler is sloped ½ in. in every direction towards the mercury bowl. As soon as all the charge is out of the pan, the settler is filled with water to within 6 in. of the top, and the agitators made to revolve at the rate of ten to fifteen revolutions a minute. When the ore is very coarse, twenty revolutions are sometimes made; but the faster it revolves the greater the loss in mercury will be. As it is desirable to have the water discharged in small streams, it is introduced from a pipe with fine holes. In the Brunswick Mill this pipe is 1¼ in. in diameter, and runs to the centre of the pan. Fine holes 1 in. or less apart are bored on the under side, so that the water entering the settler, falls in small streams, which are equally distributed over the surface. The settler can be filled as rapidly in this way as when discharging from the end of a pipe. It has been found that the fine rain, under a certain pressure, produced in this way, has a very decided effect in settling the mercury. A settler 8 ft. in diameter will weigh 7000 lb.

When the settler is full, the water is turned off. The arms of the settler are at first raised 8 in. above the bottom, and kept so for about half an hour; they are then gradually lowered until they get near the bottom, which takes about two hours. The reason why the arms are raised, is that at the time of charging the settler, the pan is running slow, and if the arms were down, they might break off under the weight of the charge. The object of revolving the arms is to keep the light particles of ore afloat, and facilitate the settling of the amalgam and quicksilver. The arms are kept revolving for at least an hour, and generally for about 3½ hours, with the settler full. During this time, the mercury collects, and a considerable part of the flour is reduced. The top discharge plug is then removed, and clean water is

allowed to flow rapidly and freely through it for half an hour, the arms being kept in motion. The next plug is then removed, and so on. The whole operation lasts from four to five hours, or the same time as the work of the pan, the work in the settler being continued so as to reach the bottom in time for the discharge of the next pan. It is necessary that the amount of water should be carefully regulated, for if the pulp is too thick the metal will not settle. If it is too thin the ore will settle with the metal, and the separation will not take place. The same is true with regard to the speed of motion; for if the motion is too rapid the mercury will remain suspended, and if it is too slow the sand will settle. The settlers should never be allowed to become clogged by the accumulation of heavy materials on the bottom, but should always be cleared if there is a disposition for them to collect there. This condition will not be helped by an excessive use of water, for after a certain amount of it has been added, even though the heavy pulp be kept in suspension, the mercury sinks to the bottom. It takes from fifteen minutes to half an hour to empty the settler. When it is discharged by the last side hole, there will be about 8 in. on its bottom. A fresh charge from the pan is run into it, and agitated for about half an hour without any water, as the hot pan charge will collect and carry the heavy sand. Water is then added, as has been determined by experience, or may be determined by a horn spoon assay. Under these conditions the settler will not clog. When the operation is properly performed there is no danger of the settler being choked with sand. The amount of water used is about 9 lb. a minute. There will be found on the bottom, when it is discharged, some coarse sand, sulphurets, fine iron from the stamps and pans, and some mercury and amalgam. There is always at least 100 lb. of mercury in the mercury receiver. The discharge of the mercury and amalgam is made either by loosening the plug in the bottom of the mercury receptacle, and allowing it to flow into an iron vessel containing some water, by dipping it out of the receptacle, or by a syphon in the bottom of it, which allows the mercury to flow continuously as it accumulates. This last is the best disposition.

In some of the modern settlers, there is no mercury bowl on

the outside, but only a slight depression on the inside of the settler, toward which the bottom slopes in every direction. In the bottom of this depression an inverted syphon is placed, from which the mercury discharges as it collects.

The settlers are thoroughly cleaned once a week, by which time 300 lb. to 400 lb. of mercury and amalgam have accumulated, all the iron being carefully scraped to remove it. When any woodwork is connected with it, the wood is never thrown away, but is always burned in a retort, in order to collect the mercury and amalgam which always penetrates the pores. There is generally one settler for every two pans.

From the settler the sluice runs to an auxiliary settler called an agitator, Fig. 162, or dolly tub, of which there are generally,

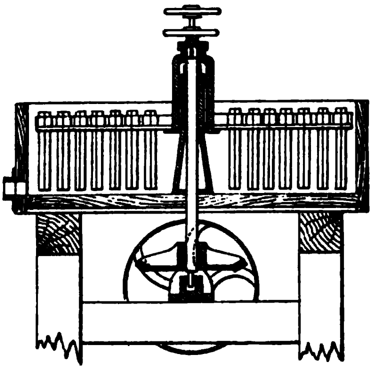


Fig. 162.

in large mills, one for every five or six settlers. They are made of wood, and are from 8 ft. to 15 ft. or 20 ft. in diameter, and from 2½ ft. to 4 ft. deep. They have four iron arms working on a cast-iron cone bolted to the bottom, with six to eight wooden uprights, which reach nearly to the bottom of the pan, and which make from ten to twenty turns a minute. A constant supply of water must be kept in the agitators. A coarse sand accumulates here, containing a small quantity of mercury, amalgam, and sulphurets, and considerable iron. The assay value of the material collected is generally less than that of the ore, but greater than that of the tails. The agitator is not now very much used except in very large mills, and in many of these some kind of concentrator, or a special settler, supposed to be specially adapted to that peculiar

ore is used in place of it. The practice in Colorado is to run the material from the agitator over copper-plated sluices, 75 ft. to 100 ft. long, having riffles every 3 ft., and then through Hendy's concentrators, which, on the whole, have not given very good results, as they are applicable to ores containing a large quantity of heavy sulphurets, and which contain so much slime, that they coat the mercury and prevent its collecting. They work successfully in California on ore adapted to them. In Nevada the tails from the agitators usually run over blanket sluices, but the material is so poor they are sometimes not washed oftener than once a week. The material discharging from the agitators is called tailings. That which discharges from the slime vats is called the battery slimes. These last, when collected, are treated in the pans like the ore. At Stewart's Mill the tails are run over riffled sluices and amalgamated plates. From 30 lb. to 40 lb. of mercury a week, which contains but very little silver, are collected in them. The tailings from this mill average 25 oz. of silver. The lowest tailings with the best working are never below 15 oz. to 16 oz. While the Stetefeldt and Airey furnaces were being used, they averaged 40 oz.; and when they were re-roasted and re-treated, only 25 per cent. of this was recovered, and as this did not pay, the furnaces were abandoned, and the reverberatory furnaces now used, built. Before this mill was burned, the Hunt and Douglas process* was introduced to save the copper, and increase the yield of silver. At the Consolidated Virginia the tails run over blanket sluices, six sluices wide, and 1100 ft. long. They have a fall of ½ in. to the foot. At the Brunswick Mill the tails run over blanket sluices which are 26 in. wide and 400 ft. long. There are eight abreast, but only seven are worked at a time. They are divided into two divisions, each 200 ft. long. They are both washed every six hours, and each has two men to do the work. The four men collect 5 tons of sulphurets in twenty-four hours. For the slimes there are eight slime vats, which are worked four at a time. The slimes are worked exactly as the ore is, but are kept separate from it, as there is not so much gold in them as in the ore. Buddles and Frue Vanners are also used to concentrate the tails. The following Table gives the total quantity of

* ENGINEERING, vol. xxii., pp. 419, 437.

material consumed in the process at the Tombstone Mill and the cost per ton of pulp treated for the year 1882-3.

	Total.	Per Ton.
Cost of fuel	$22,838.56	$1.06
Chemicals	13,624.79	0.64
Illumination	729.49	0.03
Lubrication	845.02	0.04
Castings	6,757.06	0.31
Hardware	555.62	0.03
Tools	240.42	0.01
Sundries	3,018.03	0.14
Labour	56,892.43	2.65
Total	$105,501.42	$4.91
Quicksilver, lb.	21,629	$1.00
Salt, lb.	110,094	5.12
Blue stone	22,366	1.04
Cords wood	2,526⅞	0.12
Days' labour	13,011¼	0.60

The quantity of quicksilver here given is that which was *added* during the year without taking account of what was in the pans in the beginning and end of the twelvemonth, and as there has been a reduction in quantity of ore the amount given per ton is undoubtedly too low.

All the mercury collected from the pans and settlers is carefully washed and strained through a canvas bag to remove as many impurities as possible, and is then again strained. All the amalgam collected from the shoes, dies, and elsewhere, is cleaned in a Knox pan, which is an iron pan with wooden shoes with handwheels like an ordinary pan. The amalgam is washed with an excess of mercury, and the muller is raised or lowered according as the particles must be brightened or not. They vary in size from 30 in. to 4 ft. and weigh from 1500 lb. to 3000 lb.

The bags for straining the mercury are generally made of cotton duck, either sewed directly on to an iron ring as in most of the mills in California and Nevada, or attached to leather sewed to the ring as in some of the mills in Colorado.* At the Bodie Mill the strainers are made of Canton flannel. These bags are from 10 in. to 12 in. in diameter at the top, and from 24 in. to 28 in. long. They often hold as much as 1200 lb. of amalgam. The

* Page 342.

weight of the mercury is always sufficient to make a preliminary straining without its being touched. They are set in a hole cut either in a table, Fig. 163, or in boxes prepared for the purpose, which were formerly always left open, but are now arranged so that they can be locked, Figs. 164-6, which is the best disposition.

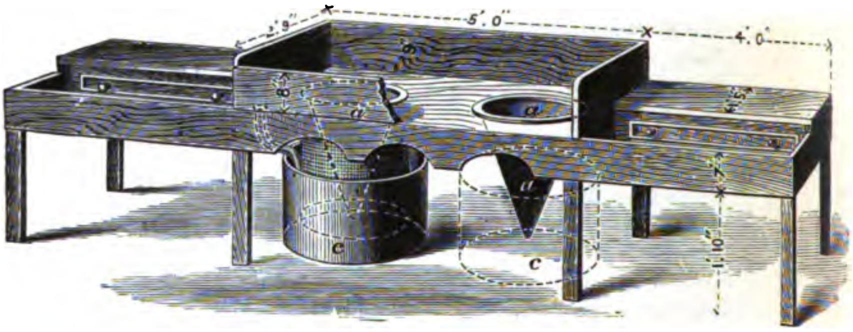


Fig. 163.

At Judd and Crosby's Mill the mercury is dipped out of the mercury receiver of the settler with a cup once in two days, as it accumulates, and the plug drawn once a week to discharge the mercury, which is carried in small pails to the strainer. In the Eureka Mill a canvas cloth curtain of the inside diameter of the tub is attached to the under side of the table, and falls nearly to the bottom of the receptacle, to insure that no mercury will be lost. At the Brunswick Mill all the mercury and amalgam from the settlers, flow into a locked tank, called a mercury safe, 6 ft. by 4 ft. and 1 ft. deep, which has 6 in. of water on the bottom. The excess of mercury flows off by the pipe *a*, Fig. 164, in the side of the bottom of the safe, and is collected in a receiver at a lower level. At this mill there is one box between every two settlers. The upper part has an opening just large enough for the men to pour in the mercury, but only the amalgamator opens the lid to work the amalgam. When the table, Fig. 163, is used the bags reach down nearly to the bottom of an iron vessel, placed on the floor, about 18 in. high and 18 in. in diameter, set inside one 20 in. high and 20 in. in diameter. These safes were formerly made of wood, as shown in

Fig. 164, but as wood is always objectionable where a large quantity of mercury comes in contact with it, the top and

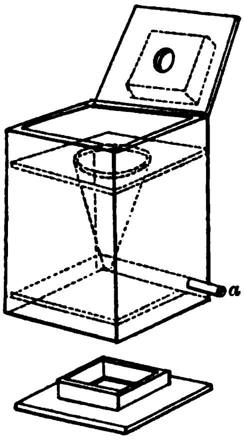


Fig. 164.

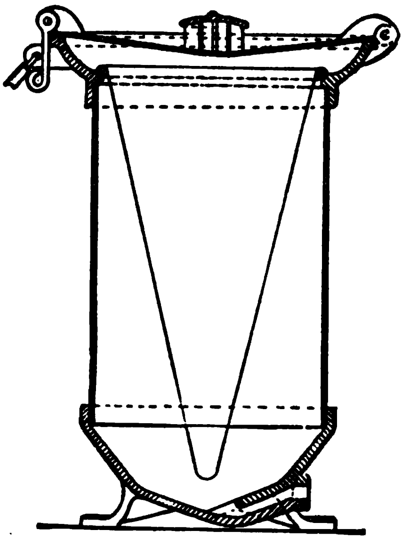


Fig. 165.

bottom are now made of cast iron, Figs. 165 and 166. The top of the safe is placed below the syphon discharge of the settler and receives through the opening in the cover, all the mercury and amalgam. The amalgam remains on the strainer. The excess of mercury runs through and flows to the mercury collecting tank. Fig. 165 shows the ordinary safe as now constructed, and Fig. 166 one for providing for the mercury when there is likely to be an excess of it by introducing a side opening *a*. All the mercury and amalgam collected in the kettles from the pans and settlers are poured into this canvas strainer, which is filled nearly to the top. In some of the Comstock mills, hydraulic strainers, with a pressure of 100 lb. to 125 lb. to the square inch, are used; they effect a great saving in fuel in the retorting. The ordinary amalgam from the canvas bags contains about seven parts of mercury for one of bullion; that from the press contains only three and a half. The practice of straining hot, which was formerly used to separate lead amalgam, is now generally abandoned.

Whatever may be the arrangements for using and collecting

the mercury, it is lost both mechanically and chemically. The whole of this loss cannot be avoided, but a very large part of it

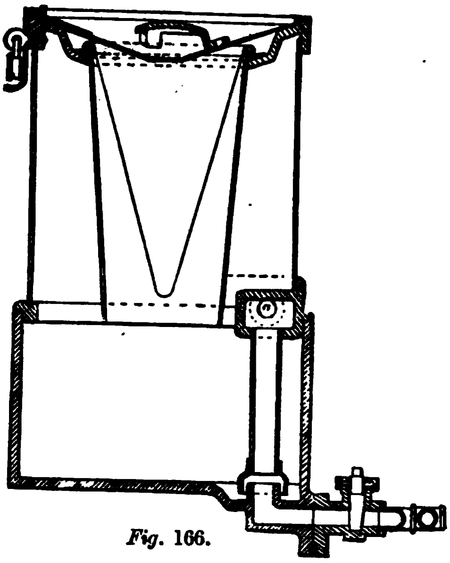


Fig. 166.

can be. By far the greater loss is mechanical. The ground under some of the old-fashioned mills was richer in mercury than a quicksilver mine. Independent of the carrying of the same mercury so often by hand, and the constant liability of loss from the careless handling of large quantities of such a heavy, volatile, and mobile liquid, which pulverises easily and attaches itself to almost everything, common prudence should protect the men from the constant danger to which they are subjected in handling it. Many devices have been invented to remedy the difficulty. The makeshifts at first adopted proved more expensive than the mercury saved by them. Mercury elevators were first used, consisting of a belt like an ordinary bucket elevator, but with cups of a peculiar shape adapted to carrying the quicksilver attached to it. This elevator was enclosed in a sheet-iron box. The lower end of this box is a tank which receives the excess of mercury from the pans at the upper part. This mercury is discharged into a tank, from which it runs to every part of the mill. Pumps have almost entirely taken its place. Two examples are given, one showing the general arrangement at the

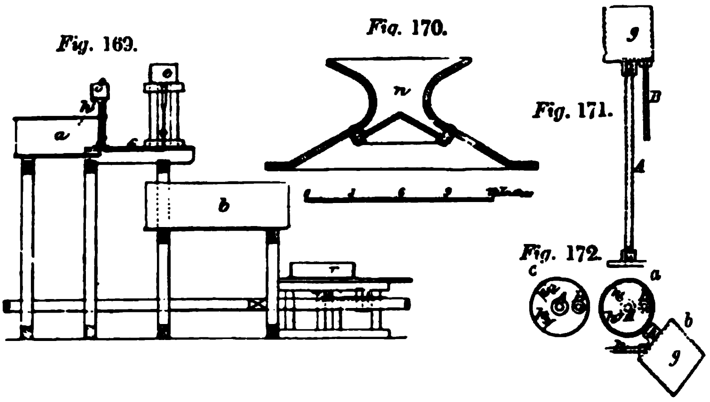
Consolidated Virginia Mill, and the other showing the details of the pumps.

At the Eureka Mill, from the bottom of the receiver an inch pipe connects the receiver boxes of the twelve bags with a cast-iron receptacle, 16 in. by 30 in. and 12 in. deep, which always has $\frac{3}{4}$ in. of water on the bottom, to prevent spattering. The tank connects by a pipe with an ordinary force pump which lifts all the mercury from the lowest level in the mill into a receiving reservoir 2 ft. by 4 ft. and 14 in. deep, which is 2 ft. above the highest point where mercury is used, which distributes the mercury to all the pans. From the main tank it runs by gravity through a $\frac{3}{4}$ in. pipe to small tanks 12 in. by 12 in., and holding from 300 lb. to 350 lb. of mercury—there being one for every two pans—each having three holes, the centre one bringing the supply of mercury from the tank, the others, one on each side, leading to the pans.

The charge of mercury is gauged by a ring which slips up and down on the circumference of the supply pipe. When the charge of pulp is put into the pan, the two side holes leading into them are closed by a plug of wood about 14 in. long, and the centre supply hole opened by the removal of its plug, until the mercury rises to the level indicated by the gauge. The supply is then cut off and the basin waits for the charge. The reservoir contains one charge at a time, and is discharged into the pans through an iron pipe.

The arrangement for moving the mercury in the Consolidated Virginia Mill is shown in Figs. 167 to 172. Fig. 167 is the elevation and Fig. 168 the plan of the pan room with the place of the pans, settlers, amalgam strainers, and cleaners. Fig. 169 shows a section of the mill on the line A A, Fig. 168. The mercury is charged from the flasks as it is received into a large tank *c*, below the lowest point in the mill where mercury is used. From this tank it is lifted by a mercury pump *d*, run by the pulley *s* to a point a little above the highest point where it is to be used, and is discharged from this tank to the various pan receivers by a pipe B shown in Fig. 171. This pipe, starting from the bottom of the mercury tank *g*, connects with the pipes *f*, which distribute the mercury from *e*, Fig. 167, to the reservoirs *g*, situated above the pans. There is one reservoir for every two pans.

This arrangement is shown in the section and plans in Figs. 171 and 172. Fig. 171 shows the section; A is the standard on



which the tank g is supported; B is the supply pipe connecting with the pipes f. In Fig. 172, a is the plan of the top of the reservoir g; b is the side elevation showing the reservoir g slightly larger at the top than at the bottom, and c is the plan at the bottom. The mercury is supplied to each of the pans by the pipes h, Figs. 167 and 172, of which there is one for each pan. The mercury and amalgam from two pans is washed into one settler b, and collects on the outside bowl, from which it flows by the pipes m, which are slightly elevated to counteract the pressure of the charge into an iron amalgam receiver n, the details of which are shown in Fig. 170. This is securely fastened on to the lid of the locked box, and below it on the inside are the conical canvas bags t. The mercury strained from them is received into the tank u, and flows from there by the pipes f to the lower reservoir c, from which it is pumped up again, and so on. When the strainers are full the amalgam is removed to be cleaned.

The arrangement of the mercury pump in most of the modern mills is shown in Fig. 173. The mercury and amalgam collects in the settler bowl A, which connects by a 1-in. pipe C with the main line of pipe D, connecting all the settler bowls. The discharge opening is 4 in. above the bottom of the bowl and 1 in. above the bottom of the inside of the pan. It is closed by a wooden plug B. The main pipe D is 1½ in. in diameter, and has an inclination of 1 in. in 4 ft. It runs to the line of strainer

QUICKSILVER DISTRIBUTION AT THE CONSOLIDATED VIRGINIA MILL.

Fig. 167.

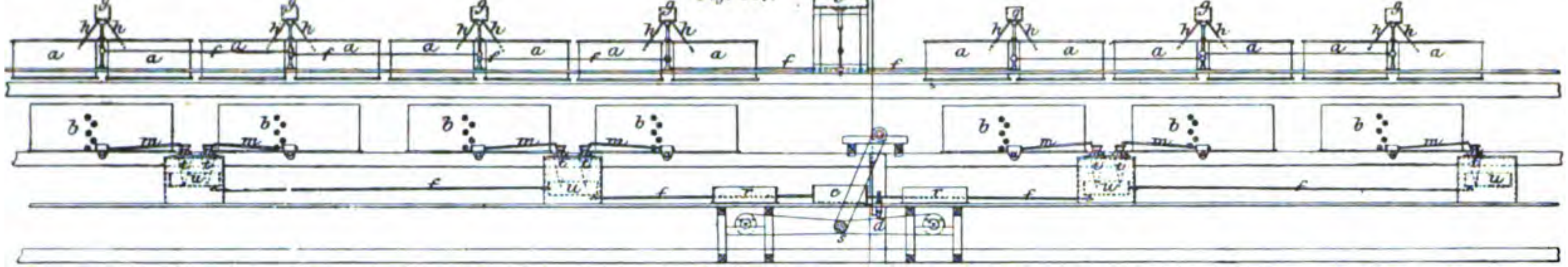
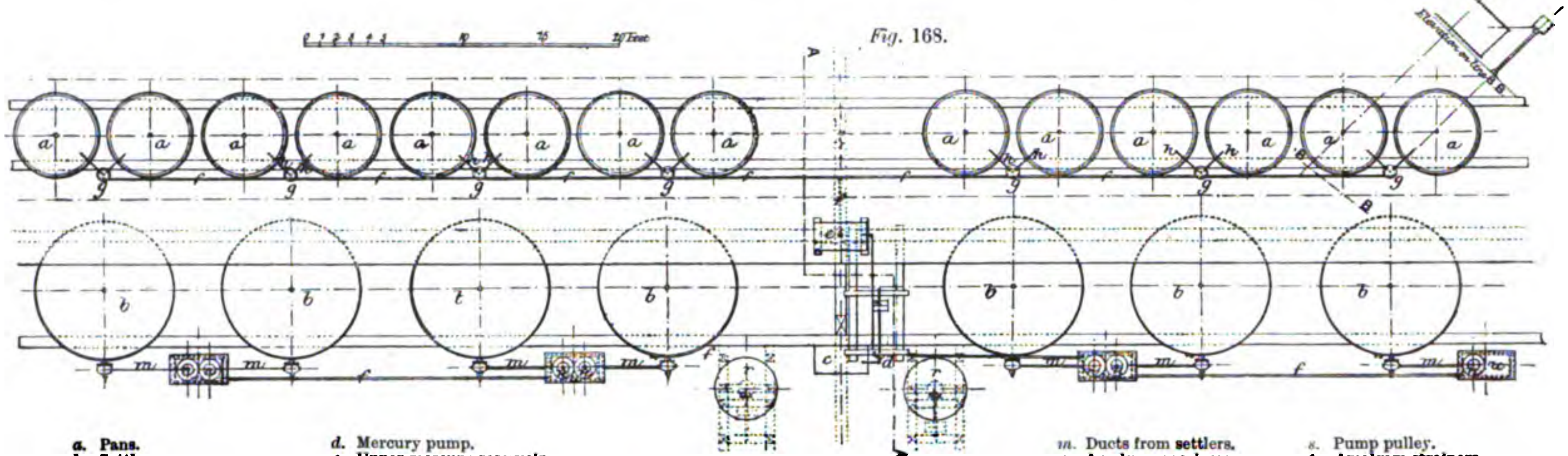


Fig. 168.



a. Pans.
b. Settlers.
c. Lower mercury reservoir.
d. Mercury pump.
e. Upper mercury reservoir.
f. Ducts from pans and settlers.
g. Pan mercury reservoir.
h. Pan supply pipe.
m. Ducts from settlers.
n. Amalgam receivers.
r. Clean-up pan.
s. Pump pulley.
t. Amalgam strainers.
u. Strainer dip box.

To face page 400]

boxes F, and connects with them by short 1-in. pipes E which project from the top of the sheet-iron lid *f*. These lids are locked to prevent their being opened except by some authorised person. At the end of the main pipe there is a small pipe *d*, which is closed with a stop-cock and is used for occasionally cleaning out the pipes when there is necessity for so doing. The strainer box F is made of wood. It is fitted to the top of the cast-iron tank G, which is 5 ft. long, 18 in. wide, and 1 ft. deep. It is made for four strainers. Its bottom is connected with the pump I by the short pipe H. The pump is placed over a drip box J, to catch any leakage which may occur. The pump discharges through the $\frac{3}{4}$-in. pipe K, which must be securely fastened on account of the jar to which it is subjected. It discharges into the upper tank L, which is a little above the highest level at which mercury is to be used. It is securely fastened on a frame and stands 20 in. above the pans. The discharge pipe from the tank L is 1 in. in diameter, and 18 in. to 20 in. long. The flow of mercury is controlled by a cock M, which is made of cast-iron and has a stuffing-box to prevent the escape of mercury under pressure. From this point to the distributing pipe N it is $\frac{3}{4}$ in. in diameter. This pipe may be laid on the floor in front of the pans and be covered with a strip of wood to prevent injury. It is connected with the pan reservoir O by a vertical pipe. There is one such reservoir to every two pans. Where the supply pipe enters the bottom of the reservoir O it is enlarged to 1 in. It is closed by a wooden plug P. The reservoir O is made of cast-iron and is placed 7 in. above the top of the side of the pans. It is supported by two 1-in. supply pipes Q, which are screwed into the flanges R, which are bolted respectively to the sides of the pan which they are to supply. The supply of mercury is regulated by marks on the inside of the tank O. When the proper height is reached the supply is cut off and the plug P withdrawn, which allows the mercury to enter the pan. The pump is made of a steel plunger and the packing of round rubber, $\frac{3}{8}$ in. thick, which is dropped into melted tallow and wiped dry to prevent the use of oil on the plunger. Below the stuffing-box and around the plunger there is a hollow space into which water rises and makes a hydraulic packing below the

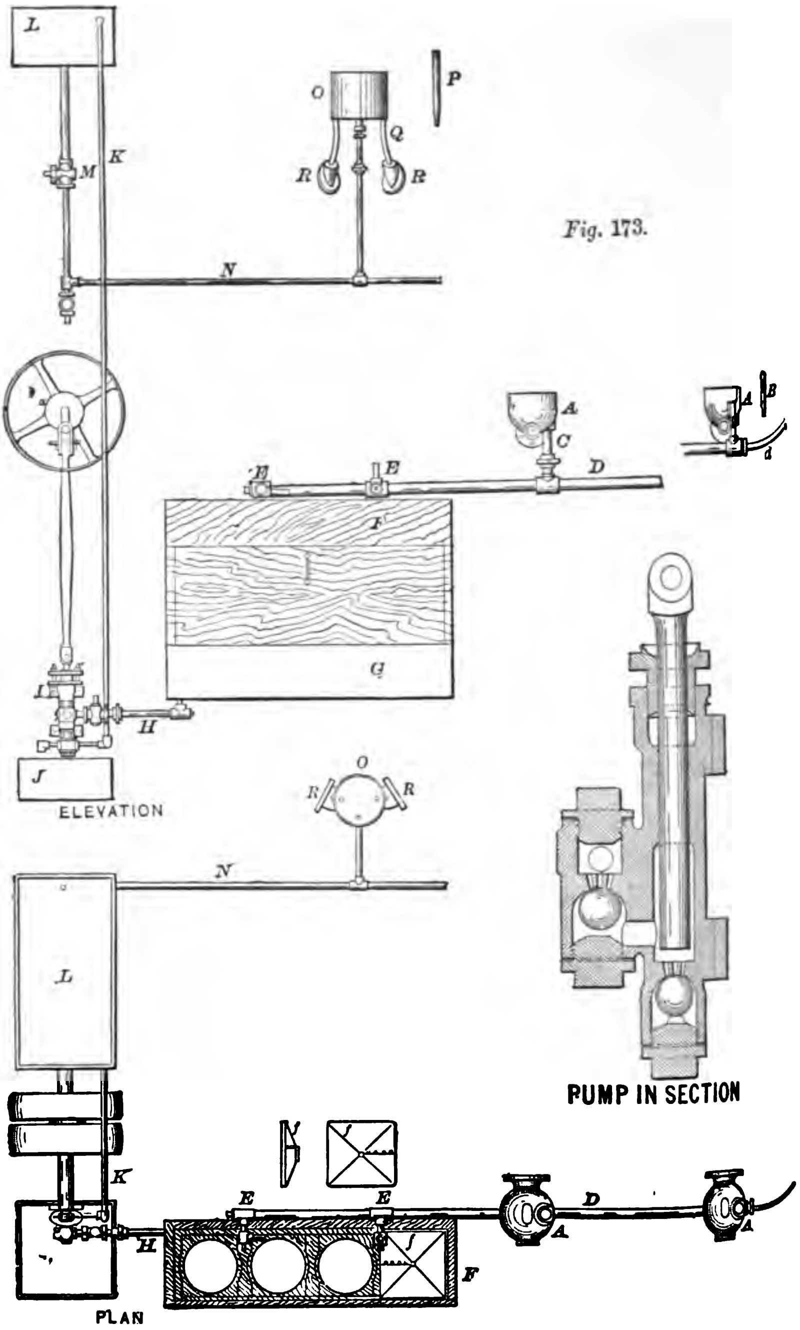

L
K
M
O
P
Q
R
R
N
A
C
D
E
E
F
G
I
H
J
ELEVATION
O
R
R
N
L
PUMP IN SECTION
K
E
E
D
H
A
A
F
PLAN

Fig. 173.

rubber one. A rubber gasket ⅛ in. thick is used on the plugs. This disposition of the pump with rubber valves, the plunger running in a hydraulic packing, effectually prevents any grinding of the mercury. The pump has a stroke of 5 in. and makes forty strokes per minute. It never must be run except when it is doing duty raising quicksilver.

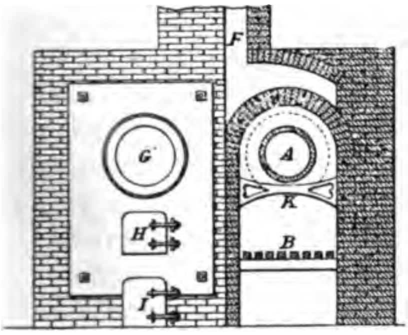
All of its movements after it is once elevated to the level of the upper receiver are automatic until it arrives again in the collecting tank, when it is again started upon its course by the pump. By this arrangement the mercury is never touched by hand after it is once discharged from the flask. It makes its round without any possibility of loss, except leakage in the joints of the pipes, which can always be guarded against. In some mills in Colorado where there is a considerable quantity of lead and other impurities in the mercury, the canvas bag is immersed in hot water, which is kept hot with steam. The surface of the amalgam is cleaned until it is quite bright. The bag is then discharged, and its contents strained a second time in cold water. It is claimed that in this way a considerable quantity of impurities, more especially lead, are removed from the amalgam, and the bullion made much finer. The loss of mercury is, however, increased, as is also the danger of salivation. The skimmings are kept separate, and either treated with acids or put on one side and treated alone when there is sufficient for treatment. During all the time the mercury is being cleaned, especially if it is heated, the weight of the metal presses a certain quantity of it through the pores of the duck, and this is caught in the receptacle below. When the single lock tanks are used, as at the Brunswick Mill, the excess of mercury runs out by a pipe *a*, Fig. 164, which is 4 ft. long, and ends in a mercury collecting tank. After the mercury is clean, and when no more runs, the bag is first squeezed with the hand, and then twisted as tight as possible. When no mercury can be extracted from it in this way the bag is lifted out on to the table, and the contents emptied. The amalgam is then worked by hand, and when no mercury can be extracted from it, it is made up into balls of convenient size to be retorted. The table where the amalgam is worked, is covered with sheet

iron. It inclines to one side, from which a pipe discharges the excess of mercury into the main pipe. It is set on an inclined floor also covered with iron, which also connects with the main pipe.

Usually all the amalgam from every part of the mill is collected and put on one side until there is sufficient to clean. The only base metals which are generally found in the amalgam are lead and copper. Iron is almost invariably found as a mechanical mixture. It comes from the small particles worn from the stamps and the pans and is not chemically combined. It is easily separated. Sulphides containing zinc, antimony, and arsenic and some other metals are also mechanically mixed with the amalgam, but they are also easily separated by the addition of mercury and treating it in the clean-up pans. The amalgam is first carefully washed with hot water to make the amalgam fluid, and thinned with more mercury, about 150 lb. being added for every 400 lb. to 500 lb. of amalgam, to free it from iron and other impurities. When there is any grease attached to it, it is carefully washed with lye. It is then, as in some of the mills in Colorado, strained through the canvas bag, or better, as is usual in Nevada, cleaned in a Knox pan. In the Brunswick Mill there are four of these, each of which is 4 ft. in diameter; at the Consolidated Virginia there are two. Here it is stirred at the rate of twenty revolutions a minute for twelve hours or less with hot water, with large additions of fresh mercury, the proportion at the Eureka Mill being equal parts of mercury and amalgam, and when quite clean is added to the mercury in the canvas bags. The hard amalgam from the pans carries a little more gold than the liquid, but it is not kept separate. At the Brunswick Mill, in the month of August, 1874, the clean up was 4110½ lb. of amalgam ready for the retort. In very large mills the amalgam collected from the pans and settlers is carried in iron cars to the retort room. The body of the car is made of sheet iron with a cover hinged at one end and provided with a lock, so that no unauthorised person can open it.

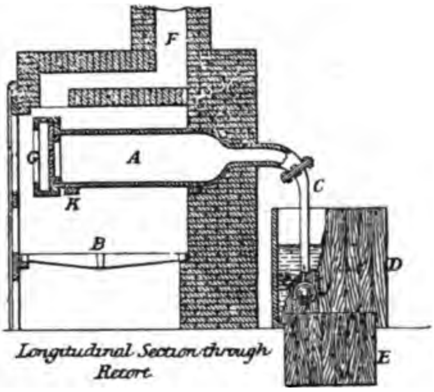
From the Knox pan there is a certain amount of sulphurets running off. They are very rich, but are not usually caught separately. They generally go to the tail sluices. At the Brunswick Mill, however, as they assay $500 to the ton, they are

caught separately, and are treated by a process very similar to the *patio* process, though differing from it in several essential particulars. They are mixed with salt and sulphate of copper, and dilute sulphuric acid. For this purpose 10 per cent. of salt and 1 per cent. of sulphate of copper is added to them, and thoroughly incorporated. They are then spread out on a flat place with an impervious floor, and sprinkled with dilute sulphuric acid, and again mixed, the acid being added until the whole is of the consistency of ordinary mortar. This is turned over two or three times a day in order to continually expose all parts of it to the air for four weeks. A ton is then charged in



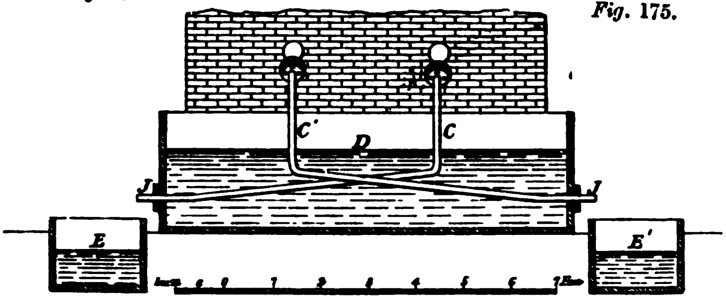
End Elevation *Transverse Section*

Fig. 174.



Longitudinal Section through Retort

Fig. 175.



Transverse Section through D and E.

Fig. 176.

the pan, and the pulp heated as hot as the steam will make it, and ground for six hours. Five hundred pounds of mercury are then added, and the operations continued for four hours more, making the whole operation ten hours. The charge is then drawn

into the settler, which is four hours in discharging, the tails going to the tails of the mill.

The retort, Figs. 174 to 176, in which the amalgam is treated, is made of cast iron. Steel retorts have been used, but they warp and burn out quicker than cast iron. It is cylindrical, about 12 in. to 14 in. inside diameter, 4 ft. to 5 ft. long, and 1½ in. thick. It weighs from 1200 lb. to 1300 lb. The front is made 2½ in. longer than the body, to receive the door. Lugs are cast in the side of this projection to catch the clamps which fasten the door G. When the door is closed and clamped, it is carefully luted with wood ashes, to prevent the escape of the mercury. The opposite end is conical in form, contracted to about 2½ in. where it connects with the exhaust pipe *c*, which is sometimes screwed into it, or, better, is attached to it by a flange. It is sometimes supported on round or square bars, but generally on pieces of iron K cast for the purpose. In many mills broken stamp stems are used for the supports. At one of the Tombstone mills these stems were so completely crystallised at the end of three years that they were useless for any other purpose without re-heating and turning. At the discharge end the retort is supported on the masonry.

The retort is set in an arched furnace, with a firegrate B underneath. The arch above the retort is open at the front, and sometimes discharges the heat into the chimney there, and sometimes returns the heat over the arch to the back of the furnace as in the drawings. The discharge pipe passes the back wall of the furnace and descends into a tank D, which is supplied with running water. These tanks are at least 2 ft. wide, and 20 in. deep, and the length of the retort furnaces, if there are a number of them, and if not, 4 ft. to 6 ft. per furnace. When there are a number of retorts, as at the Brunswick and Eureka Mills, where there are four, and at the Consolidated Virginia, where there are six, these pipes run from where they enter through the water to the opposite end of the tank, with an inclination of 6 deg. to 7 deg. The centre pipes C and C′ thus cross each other, and go in opposite directions. The mercury is condensed in the pipes and flows out from the end J which projects 6 in. to 8 in. into a basin E, containing a small amount of water to prevent the mercury from spattering.

The amalgam, generally contains about one part of silver to

four of mercury in Nevada, and one to five in Colorado. It should never be less than 1 lb. to the ounce of silver, no matter how rich the ores may have been. It is usually charged in iron trays which fit the lower part of the retort, but it is sometimes charged directly upon the bottom of the retort without any tray to hold it, but divided by discs of sheet iron to facilitate breaking up the retorted amalgam. In either case it is necessary to coat the bottom of the retort, and of the trays or pans, with levigated clay or wood ashes or whitewash to prevent the adherence of the bullion. The charge generally varies from 500 lb. to 1000 lb. At the Eureka Mill the retort is charged with 1500 lb. to 2000 lb. The fire is then gently raised and kept so until the mercury ceases to condense. From $\frac{1}{3}$ to $\frac{1}{6}$ of a cord of wood is consumed in this operation, for every 1000 lb. of amalgam treated. The time required to treat it varies from five to ten hours, depending mostly on the size of the charge. Care must be taken to increase the heat gradually at first, as too high a heat might fuse the surface of the amalgam, and prevent the escape of the mercury. At a bright cherry-red heat all but 1 to $1\frac{1}{2}$ per cent. of the mercury will be driven off. The last traces of the mercury cannot be removed except at a white heat, at which temperature the iron becomes soft and is likely to become bent and distorted, and even though it is frequently turned round, it is liable to crack, and then there will be a very large loss in mercury, which might be a serious item of expense. Besides, at such a temperature the surface of the amalgam would be likely to melt. It will, therefore, generally be best to submit to the loss of 1 to $1\frac{1}{2}$ per cent. of the mercury rather than be exposed to the risk of having a larger loss. As the retort is exposed on its whole length, it will after a time become distorted, even with the most careful use, so that it will generally be best to support it in the middle. In some of the mills two to three supports made of broken stamp stems are used, but as only one point of the bottom rests on them this does not prevent the distortion. It is better to have curved cast-iron supports as shown at K, Fig. 174, made for the purpose. In some works these supports for the retorts are built into the masonry of the sides, and in others the retorts are hung to the roof of the furnace by means

of wrought-iron straps which are supported on iron braces on the outside. In this way the retort is simply suspended in the centre of the furnace, and can be raised or lowered by nuts on the outside. If the retort becomes bulged the sling nearest the disturbed place can be raised, when the retort will resume its shape, or it can be turned round in the sling. In this way it may not only be made to last two or three times as long as it otherwise would without such precautions, which is usually not more than six to nine months, or from 150 to 300 operations, but a considerable loss of mercury may be prevented. In mills where they are not careful, several hundred pounds of mercury may be sent up the chimney through a crack in the retort, before it is discovered that the retort is out of order. It has been proposed to place condensing chambers in the flues of the retort and crucible furnaces to save the mercury in the retorted amalgam, but in most mills nothing would be saved from want of care, and it would only be an item of expense, though if carefully attended to it would more than pay for itself.

In the retorting furnace it has been almost impossible to prevent salivation, as a considerable amount of volatilised mercury remains there, which cannot be removed. When the retort gets cold condensed mercury is always found in it. In the Lexington Mill* the practice of creating a vacuum in the retort by a steam blast has been adopted. The steam and the quicksilver is condensed together. This does not make any change in the construction of the retort necessary, and the condenser is only a box made of boiler iron and perforated with tubes like a steam condenser, so as to produce the largest possible amount of cooling surface. The steam is introduced where the retort connects with the condenser. On the opposite end of the condenser is a gas pipe which connects with the front of the retort through an opening in the cover. When the steam blast is turned on a vacuum is created in the retort and pressure in the condenser. This starts the motion and the same air originally in the retort can pass in constant circulation. Its oxygen is soon absorbed by the base metals, and there remains only nitrogen. Only a

* "Mining and Engineering Journal," vol. xxxiv., p. 255.

very small amount of floured quicksilver is formed in this way. The distillation of quicksilver goes on at the rate of 600 lb. per hour with a small fire. When the quicksilver ceases to condense, the steam blast is kept up for half an hour longer, and the retort can be opened hot without any risk of producing salivation.

When the quicksilver ceases to drop into the receiver E, the retort is gradually cooled, and the retorted amalgam is withdrawn. For a charge of about 1500 lb. to 2000 lb. the firing generally takes eight hours, and consumes half a cord of wood. The amount of "retort" obtained is generally from one-fourth to one-sixth of the weight of the amalgam. At Stewart's Mill they get 1 lb. of silver from 7 lb. of amalgam. A 60-stamp mill will need at least four retort furnaces, and should have six.

The "retort metal" is a more or less spongy mass. In addition to the gold and silver, it contains lead and copper and occasionally some small amounts of iron, which are mechanically mixed with it and have not been separated in the Knox pan. When the copper contents are large, a spongy mass is formed on top of the retort metal, which is principally copper but contains some silver and lead. The copper oxidizes whenever the air reaches it. It often contains as high as 20 per cent. of silver and is exceedingly difficult to refine. A method for treating it is given in Chapter IX. The retort metal when sufficiently pure is broken up and is melted in crucibles, and cast in a cast-iron ingot mould, Figs. 177 and 178. The ingot is called a brick.

Fig. 177.

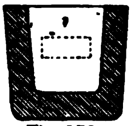


Fig. 178.

The moulds of the Consolidated Virginia are 4½ in. wide, 11¾ in. long, and 4½ in. deep. Their weight varies from 80 lb. to 100 lb. but is generally about 82 lb.

The Table on next page gives the sizes and capacity of these moulds for gold and silver.

Length.	Width.	Depth.	Capacity in Ounces.	
			Gold.	Silver
1	⅝	½	4	2
1½	1	¾	10	5
2¼	1 1/16	1	25	12
3⅜	1⅜	1⅛	50	25
3½	2	2	95	50
4½	2½	2¼	180	100
5½	3	2¾	365	200
6½	3¼	3¼	550	300
7½	3½	3¼	730	400
8	3¾	3½	910	500
9	3¾	3½	1015	600
9½	4	3½	1285	700
10	4	4	1470	800
10½	4	4	1650	900
11	4½	4	1830	1000
11	4½	4½	2200	1200
11½	5	5	2750	1500
13	6½	5½	3675	2000

The value of the gold and silver when the fineness is known can be calculated from the following data:

——	Gold in Dollars	Silver in Dollars
	$	$
1 oz. troy of gold is worth	20.6717	1.2929
1 lb. " " "	301.46	18.854
1 ton " " "	602,927.36	37,709.50
1 cubic foot troy of gold is worth . . .	361,808.64	12,355.20

In Arizona, for bullion averaging .938 fine, the loss in melting from volatilisation and skimming is 7.55 per cent. The time of melting is three hours.* The weight of the bars or bricks is 2711 oz. They require for melting 43 lb. of charcoal and 20 lb. of coke. Only 43 per cent. of the gold in the ore was saved.

Every brick must be assayed before it is sold. The amount of gold contained in the ores of the West is very variable, some containing none, others little that can be saved. It varies often in value in surface ores from 30 to 50 per cent. of the silver contained, but frequently falls as low or lower than from 20 to 25 per cent. or is altogether absent. The bricks from the Comstock mines contain from 2 per cent. to 10 per cent. of their value in gold.

* Trans. Am. Inst. Min. Eng., vol. xi., p. 105.

Some alloys of copper, lead, and silver, even after complete fusion, still retain a small amount of mercury, sufficient to make the wet assay of the silver uncertain and untrustworthy. Such alloy should be subjected to a parting process, or some such treatment as is described in Chapter IX. The usual melting furnace is described in full in the description of the United States Mint processes in Volume II.

The percentage of silver extracted is very variable, and depends upon a great variety of circumstances. These are mainly the character of the ore, the kind of work done in the mill, and the way the percentages are calculated. Sometimes the percentage extracted is calculated from the assay value of the tails, a method which can never give very accurate results. When silver ore is roasted, not more than 80 to 90 per cent. is ever extracted, though some few mills have been known to work up to 96 per cent. From free milling ores generally not much over 70 per cent. is extracted, although some of the mills in White Pine, Nevada, and Silver Reef, Utah, have worked up to 85 per cent., and those on the Comstock have been worked to a little over 80 per cent. It is not possible to fix any definite amount for the average assay value of the tailings and slimes. It varies according to the richness of the ore and the method of treatment.

At the Ontario Mill* the bullion is melted in a small reverberatory furnace with a gas generator using a mixture of charcoal and wood. The hearth is made of boiler iron with a layer of 4½ in. of Portland cement. Upon this foundation an inverted arch of firebrick is built, the mortar being made of half fireclay and half firebrick dust mixed up with a concentrated solution of borax. The sides are built in the same way. When the heat is applied the borax fuses and makes the hearth metal-tight. The hearth is 3 ft. 6 in. long, 2 ft. 9 in. wide, and 6 in. deep at its deepest part. The metal is discharged by a tap into moulds placed on a truck. It takes from five to six hours to melt and cast into bars of 100 lb.

The extraction† of the gold in ores containing silver, more

* "Engineering and Mining Journal," vol. xxxiv., p. 356.

† *Ibid.*, p. 257.

ASSAY FORM, No. 1.

PRELIMINARIES OF

SET No.______

CHECKS. Set No. 1.				CHECKS. Set No. 2.						CUPEL RESULTS.							PIPETTES.		
		GOLD.			GOLD.														
Bar No.	BASIS.	Weight.	Fineness.	BASIS.	Weight.	Fineness.	Bar No.	Furnace No.	BASIS.	Apparent.	True.	GOLD.	SILVER.		Bottle No.	Trial Weight of Sample.	Apparent.	True.	Resultant Fineness.
								1 2 3, &c.											

ASSAY FORM, No. 2.

TRANSCRIPT OF BULLION ASSAYS MADE BY THE HALE AND NORCROSS SILVER MINING COMPANY.

Set No.______ consisting of ______ Sampled ______188 Assayed ______188

PRELIMINARY.				SYNTHETICS.											PIPETTES.				REPORTABLE.			
				CUPEL RESULTS.		CORNETS.		ADDITIONS.			REMAINDERS.											
Gold.	Silver.	Bar No.	Furnace No.	Apparent.	True.	Apparent.	True.	Gold.	Checks.	Silver.	Gold.	Silver.	Bottle No.	Test Weight of Sample.	Apparent.	True.	Thousandths of Silver and Mercury.	Mercury.	Gold.	Silver.		NOTES.
			2 1 3, &c																			

ASSAY FORM, No. 3.

MEMORANDUM OF BULLION OF THE CONSOLIDATED VIRGINIA MINING COMPANY, ASSAYED AT THE CONSOLIDATED VIRGINIA ASSAY OFFICE.

Virginia, Nevada, ______188

			WEIGHT.				FINENESS.					
			BEFORE MELTING.		AFTER MELTING.							
	NAME OF MILL DEPOSITING.	NO. OF THE BAR.	Ounces.	Dec.	Ounces.	Dec.	GOLD.	SILVER.	GOLD VALUE.	SILVER VALUE.	TOTAL VALUE.	VALUE OF SAMPLES.
1 2 3, &c.												

especially when they have been roasted, is very incomplete, and the loss is very large and is greater the longer the time and the higher the heat that the ore is roasted, so that ores containing gold must be roasted at the lowest possible temperature. The loss is also greater the larger the amount of copper contained in the ore. In the Nurphy Mine, Ophir Cañon, Nevada, the amount of gold extracted was only 25 per cent. Afterwards, by the introduction of the Stetefeldt furnace, 80 per cent. was obtained. At the Alice Mill, Butte City, Montana, the percentage was only 55 per cent. At the Lexington Mill, in the same town, it varies between 58.8 and 60.8.

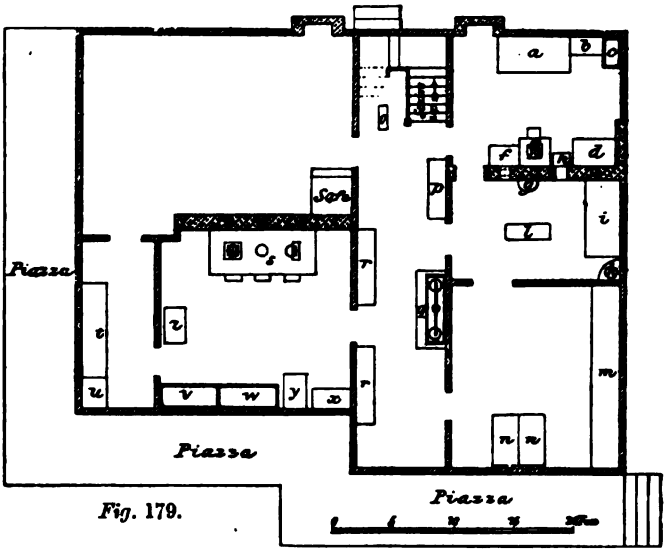


Fig. 179.

a. Ore assay table.
b. Anvil table covered with copper, for buttons.
c. Sink.
d. Ore assay furnace.
e. Still and sand bath fire.
f. Muffle furnace.
g. Distilled water tank.
h. Sand bath for parting.
i. Gold-collecting table.
k. Wash basin.
l. Work bench covered with copper, for buttons.
m. Balance table.
n. Desk.
o. Rolls.
p. Sample weigh table.
q. Bar scales.
r. Stamping table.
s. Bullion furnaces.
t. Table for smoked moulds.
u. Furnace for smoking moulds.
v. Cleaning tank.
w. Cooling tank.
x. Writing desk.
y. Platform scales.
z. Table for casting the bullion.

In order to ascertain the fineness of the bullion two assay samples are taken from each brick by boring holes into both the top and the bottom. These are enclosed in a printed wrapper with the same indications on the inside and outside of the wrapper. The label on the inside has a margin to give the size that the paper is to be folded. Both papers are exactly alike, except that one has the bottom and the other the top sample printed on them.

Bottom Sample.

From Bars No. ______________________________

Worked by ______________________________ *Mill.*

Sampled ______________________________ 188

It is then assayed; the results are recorded on the blanks Nos. 1 and 2, p. 412, and sent to the head office, a record being previously entered on the books of the assay office. The Consolidated Virginia uses blank No. 3.

The samples are kept for a year, and if after that there is no dispute about the bars, the silver of the sample is melted. The plan of the assay office of the Consolidated Virginia Mill is shown in Fig. 179. The house is small, but the office is otherwise well arranged.

The name of the mill and its value in gold and silver are stamped on each brick. It then goes to the market and passes current. Fig. 180 shows one of the Consolidated Virginia bricks.

Fig. 180.

In smelting, the loss per weight is generally 2 or 3 per cent., as it is impossible to get out all the mercury except at a very high heat, which it is not safe to do in retorting. The quality of the bullion produced is very variable, it being from 0.050, as was the

case with some of the bullion at the Meadow Valley Mill in Pioche up to 0.999. Usually the bullion does not run below 0.990, as in the case of the Comstock mines of White Pine in Nevada, and Silver Reef in Utah. It is very rarely as high as 0.999. No special effort is ever made to reach this limit, as such fine bullion is rarely ever wanted in the market. The fineness of the bullion depends generally upon the character of the ore, but sometimes is lowered by the method of working. Some of the very base ores in which most of the silver is in the form of chloride can be made to produce bullion 0.900 fine by amalgamating cold or by using a very small amount of salt with little or no copper sulphate. Generally, however, with such ores it is best to keep the bullion from 0.600 to 0.700, for although the silver chloride is amalgamated almost at once, the silver sulphide minerals remain unamalgamated with the base metal compounds. The value of the metal depends upon its fineness. As a general thing in Nevada, it will be from $1.75 to $2.00 per oz., from two-thirds to one-third of the value being gold, and from one-third to two-thirds silver.

At the Eureka Mill the loss of mercury is about 1 lb. to every 10 lb. of amalgam. The retorting is done every other day. The crude bullion or the retort metal produced in August, 1874, amounted to 8322 lb.; the melted metal was 985 fine. The value of the bullion was $406,337.12. A single deposit from a clean up made in the same month was $92,917.26, of which $63,914.67 was gold, $29,004.25 was silver. This was from six days' work in the mill. The slimes collected in the vats assayed two-thirds of the value of the ore. There is less gold in the slimes than in the ore. The mill always works up to 75 per cent. of the yield of the ore. When it works under 65 per cent. a charge is made to the mill, and deducted from the wages of the men, as it denotes carelessness or dishonesty on their part. In August, 1874, 5900 tons were treated up to 84 per cent.

At the Brunswick Mill the bullion is 994 fine; at the Consolidated Virginia, 937; at Stewart's Mill, where the ore contains much base metal, the silver is 750 to 850, the base metals being copper and lead; at the Nederland Mill it is only 700 fine, the impurities being also lead and copper.

The Table below gives the production of the Consolidated Virginia for three months of 1875:

1875.	Number of Tons Stamped.	Value.	Yield.	Profit.
		$	$	$
March	...	...	1,707,54C	
April	6565	...	670,000	31,000
May	6782	12,490	720,000	36,000

The clean up for the month of August, 1877, was $1,445,780.31. The clean up of the California Mine was $1,403,296.45, or for both mines $2,849,076.76.

The total loss in mercury in the mills of the West is very variable, and it is always slightly greater in cold than in warm weather. In some of the most carefully conducted mills it has been reduced to ½ lb. per ton, and has been in the same mill, when constant attention was not given to the matter, as high as 6 lb. per ton of ore. In Nevada it may be said to be between 1½ lb. to 3 lb. At Stewart's Mill it is 2 lb. per ton of ore, at the Nederland between 1 lb. and 2 lb., at the Brunswick Mill in 1874 it was 2 lb., and at the Eureka Mill in 1873, 1½ lb. per ton of ore treated. It is not always easy to see where this loss occurs in the modern mills, except in the retort, but there are a thousand ways of losing it. When the ores contain lead carbonate, part of this waste is in the settler, where the fine mercury is held down in the heavy materials on the bottom. When they contain copper or lead sulphide, or an excess of copper sulphate is added to the charge, alloys are formed, which although pasty, are easily floured, and thus lost. In such cases sodium amalgam may be used to advantage. Some of the mercury is lost as chloride, but the most of this salt is reduced by the iron. Mercury is a much more volatile metal than is usually supposed, and cannot be exposed, unless under cover of water, without some volatilisation. If any of it is dropped it is impossible to pick it all up, as it breaks and some of it becomes scattered as an impalpable powder. In the Owyhee Mill globules of mercury were found in the dust of the cross timbers of the roof, 40 ft. to

50 ft. above any place where it was used. It could never have reached there except in a state of vapour or of exceedingly fine dust. So true is this that in industrial establishments or laboratories which are dry, and where it is likely to drop, or much of it must be left without any cover, it is usual to sprinkle the floor with flowers of sulphur. This precaution is not necessary where the floors are constantly wet, as the water keeps down the dust and helps to carry off the globules.

All the mercury of the large mill will be moved at least forty to fifty times a day. In a 60-stamp mill, crushing 3 tons to 4 tons per stamp a day, and working $125 to $150 ore, the total weight of all the mercury moved in twenty-four hours will not be less than from 30 tons to 35 tons. The value of the mercury in rotation often represents a capital of from $30,000 to $40,000. When this is moved, as it formerly was, in pails and buckets, and generally handled by the men, it was impossible to avoid loss. Add to this the flouring produced in all the processes of stamping, grinding, and retorting, and there accumulates in the mere mechanical ways a large number of very small losses which in the aggregate make a very large amount, of which that lost as flour is no inconsiderable part. Other mechanical losses are caused by grease of any kind, and by the hydrated silicates of alumina and magnesia, both of which have what is known as the soapy feel, and coat mercury like grease. Experience has shown that as soon as the mercury becomes dirty, either mechanically or chemically, it not only loses its power to amalgamate, but is also much more readily floured than if it was clean. There are, besides these, chemical losses owing to the reactions of the pulp on the mercury. Add to this carelessness on the part of the men in charging it into the pans, and it is no longer a matter of surprise that in one of the mills of Colorado, more than 1000 lb. of it were found in the foundations under the pans, and that the loss of mercury varies from ¾ lb. to 6 lb. or 8 lb. per ton of ore treated. All the chemicals, if carelessly used, are likely to produce a certain loss. The sulphurets of copper and the chlorides of silver are decomposed with the formation of soluble salts of mercury, which can be precipitated in the laboratory, from the water of the pans

and settlers. The chlorides precipitate their metals, forming an amalgam with the silver, and do no harm; but the copper, if in excess, not only lowers the grade of the bullion, but as it so easily oxidizes, makes the mercury dirty. Manganese causes a large loss also. In fact, almost any substance in the ore capable of being decomposed by the mercury directly or indirectly, will occasion loss. Too little attention is paid in certain localities to the battery water, which if impure may alone cause a considerable loss. In order to prevent the loss being produced in the pan, the chemicals should be made to produce the decomposition of the ore. The introduction of cyanide of potassium towards the close of the pan process will help to collect the flour and clean the mercury. In order still further to reduce the loss, no substance that has come in contact with the mercury should be allowed to go out of the mill. All the wood and ironwork of the mill which has ever come in contact with mercury should be retorted. The woodwork of the pans and settlers, and all the wooden vessels, as pails, &c., and also the floors on which they are set when the mercury is not moved by machinery, should be distilled. The charcoal should then be ground and added to the pulp in the pan, as the ashes will contain silver and gold if the wood has contained amalgam. The cloths and canvas bags used as strainers should also be retorted. Every piece of cast-iron should be examined for holes, and if there are many, it should be retorted, as should all the copper plates, which should afterwards be melted. This will effect a saving in mercury, and a small saving of silver. By using all these precautions there will still be a loss, but with proper care and intelligence it may always be kept at a minimum. On account of all these losses, precautions are taken against them in all the best modern mills. Those against mechanical loss consist in appliances for mechanically handling the mercury, which have been described above, and the addition of certain materials to neutralise the effects of substances likely to coat it.

The efforts to reduce the loss in mercury have led to the invention of machines for catching it. Among them is one* which was formerly in use in some of the mills below Virginia City.

* Invented by Mr. Varney.

The machine consists of two cones revolving the one over the other. The inside cone is stationary, and is 2 ft. high. The sides turn up to the same height below the bottom and form a cup in which mercury is placed. The cone part is covered with amalgamated copper plates. The outside cone revolves, and is 3 ft. high on its inside. Eighty shoes made of hard rubber or wood 8½ in. long, 2 in. wide at one end and 3½ in. at the other, and 1½ in. thick, are placed in such a way that in the revolution of the outside, every spot of the entire height of the amalgamated plates on the inside cone will be rubbed. These shoes are dovetailed into the cone in such a way as to be easily removed for repairs. In the lower part of the inside cone, 300 lb. of mercury are placed so that it is just below the level of the amalgamated plates, and almost touches the lower ends of the revolving cone. The outside cone revolves at the rate of twenty-five to thirty times a minute. The tailings of the mill are discharged into the machine, and pass between the two cones down to the surface of the mercury, and out at the side. The shoes of the revolving cone keep the amalgamated plates bright, and the mercury in the tailings is sufficient to keep them amalgamated. The finely divided amalgam is caught on the amalgamated plates; the floured mercury coming in contact with the surface of the mercury in the bottom of the pan, which is always in motion, is caught by it. In one of the mills where this machine was first introduced, before it was perfected in its details, half the product of twenty stamps, each crushing 3 tons a day, caught 28 lb. of mercury in forty-eight hours, and in the next sixteen days it saved only 59 lb. of mercury and 8 lb. of amalgam, the small quantity being caused by the fact that from defect in construction, the shoes were so near the copper plates as not only to rub off the amalgam, but also to wear the copper. Such devices as these are now, however, generally abandoned for regular concentration. That much is to be gained by passing the whole of the tailings through concentrating machines is shown from the fact that a 20-pan mill without any special arrangement for catching the mercury, working on tailings, made a gain of 700 lb. of mercury in six weeks, which was but a small part of the mercury actually contained in the slimes.

The material which is concentrated at the Tombstone Mill consists of sands, pan slimes and battery slimes, mixed. The battery slimes consist of that portion of the ore which is stamped so fine that it will not settle in the tanks. This portion of the ore has always escaped treatment and has been a loss hitherto; in the new treatment it is intended to save it all. The pan slimes are formed by the grinding in the pans. They have lost the greater portion of their silver by amalgamation, but are too fine to work well on the concentrating tables. Experiments already made show that they can be successfully concentrated in buddles. It is not necessary to carry this concentration very far, as a certain amount of the clayey slimes are needed to give the sandy concentrates the proper consistency for holding together in a subsequent furnace treatment. About 55 per cent. of the value of the tailings was obtained in the concentrates. This was subsequently materially increased by separating the battery slimes from the tailings, and by concentrating the pan slimes by themselves. The result of these trials shows that in a mill built for the work, at least 75 per cent. of the silver in tailings can be recovered, when the ores contain from 4 to 6 per cent. of lead; the material treated being the tails which have laid waste for years. The method of operation is as follows:

From the settlers the tailings run to an agitator, the purpose of which is to make the supply to the tables constant; thence to Frue Vanners, where the valuable portion is separated and the tailings run to a receiver. These tailings are then raised to a water separator, where the fine slimes are separated from the sands, and the former are concentrated in buddles.

Occasionally the floured mercury and amalgam collects in pockets in the reservoirs. From 15 lb. to 20 lb. have been taken at one time from such a pocket. The failure to collect the mercury lost is one of the very weak sides of pan amalgamation.

Sometimes on the Comstock, where the pan covers are very tight, an explosive gas is formed to such an extent that when the charge in the pan is left for some time it accumulates in such quantities as to cause an explosion when a lighted candle is brought near the charging hole. It has been suggested that

under these circumstances hydro-carbons form in the pan,* but nothing is really known as to the composition of the gas. Such accidents are reported only on the Comstock.

In the whole western country the winters are very severe, and require houses of special strength to protect both the men and the machinery against cold and wind. The construction of these buildings has been shown in Figs. 128-137. As much of the process as possible must be conducted under a roof, so that the buildings must cover a large extent of ground, and as there must be a considerable fall for the purposes of the process, they must be high. The Lexington Mill,† Butte City, Montana, is 326 ft. long and 139 ft. deep. The area covered is more than 29,000 square feet. The total number of square feet occupied by the mill is 45,000. From the charging floor of the dry kilns to the boiler room the fall is 61 ft.

The buildings of the Lexington Mill are arranged as follows:

	feet.
Dry-kiln buildings	120 by 27
Stetefeldt furnaces	103 ,, 47
Pan room	163 ,, 35
Engine and boiler room	70 ,, 43
Battery room	120 ,, 32½
Firemen's floors, each	33 ,, 13
Agitator room	24 ,, 22
Bullion furnace	35 ,, 30

All well-constructed mill buildings are very expensive, and must be kept in the best repair.

The charge for milling in Stewart's Custom Mill in Colorado is 35 per cent. of the assay value of the ore. On the Comstock the price is generally settled by contracts which have several years to run, and is a matter of special arrangement between the mill and mineowners.

The yield of the ores is very variable, as is also the percentage of extraction. The ores which can be most cheaply worked are those which contain the silver chiefly as chloride. Some of the best instances of such free milling silver ores are those of White Pine, Nevada, and Silver Reef, in Utah. When the ores contain the silver chloride mixed with lead carbonate and various copper,

* Berg Hut. Zeit., xxx., 1871, p. 66.

† "Engineering and Mining Journal," vol. xxxiv., p. 255.

arsenic and antimony minerals, the working becomes more difficult. Such is the case with the ores of the Comstock, which, although they contain very small amounts of base metal, have a large part of the silver in the form of sulphide, and a part of the gold in iron pyrites. This ore mills generally from 70 to 80 per cent. The most difficult class of ores to work are those which contain the complex silver sulphides combined with other metals. It is almost always necessary to roast these ores, notwithstanding which, those that contain quartz as a gangue with silver sulphide and a little of the iron and copper pyrites, can be roasted and worked at a cost not much above that of the baser free-milling ores, and at least 90 per cent. of their assay value can be extracted from them. Such ores are not often met witb. The larger part do not contain enough sulphur to make the chloruration complete, or contain such large quantities of the sulphides of zinc, antimony and arsenic, that they can only be roasted at a great expense of time and money. Such ores as these are found at Morey, in Nevada, and also in a number of places in Arizona. It is often very difficult to decide whether an ore can be best milled or smelted, and, if milled, whether it requires roasting. The very frequent mistakes in deciding upon the proper plant and process to work such ores have scattered over the country idle reduction works which are going to ruin on account of an improper method having been selected to work the ores. Such failures are rarer now than they formerly were, more skill and experience being required by the owners for the erection of a plant than was formerly the case.

At the Tombstone Mill, in Arizona, one ton of ore yielded on the average:

		1882.	1883.
		$	$
Of fine silver, 41 ounces	... market value	45.60	34.92
,, gold, 0.084 ounce	... ... value	1.74	1.74
Total market value per ton	... ...	47.34	38.64
,, assay value per ton	... ...	54.76	
Percentage extraction of silver	... ...	76.76	8.00
,, ,, ,, the gold	... ...	46.67	5.00

The following Table gives the variation of the ores for several periods:

	Gold.	Silver.	Total Value.
	oz.	oz.	$
1. June-September, 1879, per ton ...	0.140	63.81	85.39
2. October-March, 1879-1880 ,, ...	0.121	·36.80	50.08
3. April-September, 1880 ,, ...	0.103	54.13	72.14
4. October-March, 1880-1881 ,, ...	0.117	50.71	67.98
5. April-September, 1881 ,, ...	0.104	49.66	65.35
6. October-March, 1881-1882 ,, ...	0.070	34.79	46.42

The cost of treatment to the large mill companies on the Comstock Mill will not usually be as high as $8.00. At the Brunswick Mill in January, 1874, it was $7.41. The average for the whole year was a little higher, being $7.76. They rarely charge less than $12.50 on large and long contracts. The price in different localities will vary according to the number of mills in the district. Usually in custom mills it is the practice to charge from one quarter to one half, or even more than the cost of treating the ores, as a profit. An exception is made in the case of custom mills which own mines which they lease to the miners, expecting to make the profit out of the mine and not out of the mill. But this is unusual. Generally gold mills do not guarantee any percentage of yield. In silver mills, where the ore is worked raw, they guarantee generally from 70 to 80 per cent. of the assay value. In some cases the total clean-up of the mills is given to the mine, that is, after working the ore the entire product is collected and given to the ore owners, who pay the charges for working. When the ore requires to be roasted, the mill will generally guarantee 80 per cent. Sometimes the mills, as at Austin, guarantee 80 per cent. only on ores which assay over a certain fixed value. Whenever a percentage is guaranteed, the tailings and slimes belong to the mill company, but when the ore has been worked on the clean-up plan the tails belong to the mining company, unless a previous arrangement has been made with regard to it. In purchasing ores the cost of milling is always deducted from the assay value. When custom ore is worked, the mine generally pays for hauling the ore to the mill, whether it is transported by pack animals, wagons, tramway or railroad. The actual cost of milling ores in wet crushing varies from $4 to $8. In dry crushing it is from $6 to $12. When the ores require roasting it varies from $12 to $28. In some few

cases these figures are exceeded, and sometimes it is possible to work at a less cost than the least of these prices. The great variation in the ores and facilities for working, and in the methods of extracting the precious metal, make it impossible to

PACIFIC MILL AND MINING COMPANY.

Operations of the __________ *Mill, for the Month of* __________ 188 .

SUPPLIES ACCOUNT.

	DEBIT ENTRIES.					CREDIT ENTRIES.						
—	ON HAND. ——1st, 1874.		Voucher Number.	PURCHASED.	TOTAL RECEIPTS.	SOLD.	ON HAND. ——1st, 1874.		CONSUMED.			
									PER TON.		TOTAL.	
	Quantities.	Cost.		Cost.	Cost.	Cost.	Quantities.	Cost.	Cost.	Quantities.	Quantities.	Cost.
		$ cts.		$ cts.	$ cts.	$ cts.		$ cts.	$ cts.			cts.
Wood	Cds.						Cds.			Cds.	Cds.	
Quicksilver ..	lb.						lb.			lb.	lb.	
Catings ..	lo.						lb.			lb.	lb.	
B. Stone ..	lb.						lb.			lb.	lb	
Miscellaneous												
Totals ..												

Total cost of supplies used .. ..

GENERAL EXPENDITURES.

		dols.	cts.
Labour	as per pay-roll		
Freight of ore..	do. bill		
Water	do. do.		
Hardware	do. do.		
Lumber	do. do.		
Repairs current	do. do.		
Repairs extraordinary ..	do. do.		
Team expenses	do. do.		
Miscellaneous..	do. do.		
	Total cost per ton		
	Total cost for tons		

		dols.	cts.
GROSS RECEIPTS:			
	Earned by working tons ore		
	Proceed bar , slime bullion		
MILLING COSTS:			
	Supplies consumed		
	General expenditure		
	Profit		

CASH.

		Dollars.	Cents.	Dollars.	Cents.
RECEIVED					
	From Pacific Mill and Mining Company (borrowed)..				
	From sale slime bullion				
	From Mining Company ..				
DISBURSED					
	To Pacific Mill and Mining Company (dividend) ..				
	For general expenditures..				
	For purchase of supplies				

make any general statement as to the cost of the process, as the cost varies very frequently in the same district.

When the supply of ore is short, and the number of mills large, there have been times when custom mills have paid more for the ore than it was worth, in order either to get the monopoly, or to keep the mill running, as it is always very expensive to shut down. As a usual thing, however, it is the miner and not the millman who suffers, for there are generally fewer mills than the mining capacity of the district needs; hence the miner is at the mercy of the millmen. The blank form, p. 424, gives the methods of keeping the account to get at the exact cost of milling of the Pacific Mill and Mining Company.

The first Table below, which was prepared by Mr. M. R. Elstner, superintendent of the Brunswick Mill, gives the running expenses of this mill for the year 1873. The second Table, p. 426, also prepared by Mr. Elstner, gives the total expenses for the month of January, 1874, and the third Table, p. 428, also prepared by Mr. Elstner, gives the record of the bullion deposits of November, 1874, at the United States Mint in Carson City.

BRUNSWICK MILL EXHIBIT FOR THE YEAR 1873.

STOCK SUPPLIES.

On Hand 1st of January, 1873.			Received during the Year 1873.		Consumed during the Year 1873.		On Hand Close of the Year 1873.	
		dols.		dols.		dols.		dols.
Wood cords	..	..	993$\frac{5}{16}$	9,937.10	971$\frac{4}{15}$	9,717.10	22	220.00
Quicksilver lbs.	10,950	9,855.00	58,068$\frac{1}{2}$	58,105.57	56,732$\frac{3}{4}$	54,216.17	12,281	13,744.40
Castings lbs.	136,653	9,565.90	639,783	41,539.96	677,300	44,976.97	99,136	6,128.89
Sul. copper lbs.	5,024	629.87	39,396	4,741.99	42,420	5,131.86	2,000	240.00
Lard oil.. gals.	..	..	300	675.00	300	675.00		
Coal oil.. gals.	..	..	890	534.00	890	534.00		
Miscel. sundries ..	..	..	..	23,101.97	..	23,101.97		
Hauling and weighing ore tons	273^{1570}	558.50	34,226^{330}	69,690.60	34,499^{190}	70,249.10		
Totals.. ..	..	20,609.27	..	208,326.19	..	208,602.17	..	20,333.29
Received ..	..	208,326.19				Consumed ..		208,602.17
		228,935.46						228,935.46

BULLION MEM.		ORE.			LABOUR.	
Date.	Pounds.	Crown Point Mine.	Tons.	Pounds.	Milling Ore.	——
						dols.
For 1873	60,206$\frac{3}{4}$	On hand Jan. 1, 1873 ..	273	1570	Pay roll	43,889.00
		Received during 1873 ..	34,226	330	Boarding-house bills ..	15,342.90
		Total	34,499	1900		59,231.90
		Worked	31,499	1900		
		Balance				

LEDGER ACCOUNTS.

	Face of the Ledger Last of the Year 1873. Dr.	Face of the Ledger Last of the Year 1873. Cr.	Balances of Ledger Last of Year 1873. Dr.	Balances of Ledger Last of Year 1873. Cr.	Mill Account for the Month. Total Expense.	Mill Account for the Month. Expense per Ton.	Mill Account for the Month. Total Credit.	Face of the Ledger 1st day of January, 1874. Dr.	Face of the Ledger 1st day of January, 1874. Cr.
	dols.	dols.	dols.	dols.	dols.	dols.	dols.	dols.	dols.
Wood cords	9,937.10	220.00	9,717.10	..	9,717.10	.28	..	220.00	
Quicksilver lbs.	67,960.57	13,744.40	54,216.17	..	54,216.17	1.57	..	13,744.40	
Castings lbs.	51,105.86	6,128.89	44,976.97	..	44,976.97	1.30	..	6,128.89	
Sulph. copper lbs.	5,371.86	240.00	5,131.86	..	5,131.86	.15	..	240.00	
Lard oil gals.	675.00	..	675.00	..	675.00	} .03			
Coal oil gals.	534.00	..	534.00	..	534.00				
Miscellaneous ..	23,101.97	..	23,101.97	..	23,101.97	.67			
Hauling and weighing	70,249.10	..	70,249.10	..	70,249.10	2.04			
Stock	..	20,609.27	..	20,609.27	..	..	..	..	20,333.29
Nevada Mill Co. ..	418,946.24	268,029.94	150,916.30						
Mill	..	418,474.39	..	418,474.39	..	..	418,474.39		
Inventory	20,333.29	..	20,333.29						
Labour	59,231.90	..	59,231.90	..	59,231.90	1.72			
	727,446.89	727,446.89	439,083.66	439,083.66	267,834.07				
		Net credit	..	..	150,640.32				
					418,494.39	7.76	418,474.39	20,333.29	20,333.29

MEMORANDUM.

	dols.
Stores, &c., January 1, 1873	20,609.27
Profit and loss	150,640.32
	171,249.59
Surrender of Brunswick Mill Company's account ..	150,916.30
Opened new accounts with stock	20,333.29

M. R. ELSTNER, Superintendent.

BRUNSWICK MILL EXHIBIT FOR THE MONTH OF JANUARY, 1874.

STOCK SUPPLIES.

	On Hand First of the Month.		Received during the Month.		Consumed during the Month.		On Hand Close of the Month.	
		dols.		dols.		dols.		dols.
Wood ..cords	22	220.00	$122\frac{7}{8}$	988.75	$144\frac{7}{8}$	1,208.75		
Quicksilver lbs.	12,281	13,744.40	6,398	7,677.60	7,492	7,927.60	11,187	13,494.40
Castings lbs.	99,186	6,128.89	68,556	3,941.97	90,682	5,440.92	77,000	4,629.94
Sul. copper lbs.	2,000	240.00	5,690	686.62	6,200	757.00	1,490	169.62
Lard oil gals.	..	..	40	50.00	40	50.00		
Coal oil gals.	..	..	80	48.00	80	48.00		
Miscel., sundries	..	..	..	2,038.43	..	1,438.43	..	600.00
Hauling & weighing ore tons	..	..	$4,285^{470}$	8,741.88	$4,285^{470}$	8,741.88		
Totals.. ..	..	20,333.29	..	24,173.25	..	25,612.58	..	18,893.96
Received ..	..	24,173.25				Consumed ..		25,612.58
		44,506.54						44,506.54

BULLION MEM.		ORE.			LABOUR.	
Date.	Pounds.	Crown Point Mine.	Tons.	Pounds.	Milling Ore.	—
						dols.
Jan. 9	642	Received	4285	470	Pay roll	4329.70
„ 12	755				Stadtmuller & Co.'s bill	1045.83
„ 15	674				Beer and Brothers'	669.22
„ 19	809				Stadtmuller account, Branch.. .. „	78.25
„ 21	512					6123.00
„ 26	685					
„ 29	540					
„ 30	732					
Feb. 3	1244½*	Total	4285	470		
		Worked	4285	470		
Total ..	6593½	Balance				

* This 1244½ lbs. from the cleaning up of mill from January. A close clean up of mill, at the close of every month is the custom.

LEDGER ACCOUNTS.

	Face of the Ledger Last of the Month.		Balances of Ledger Last of the Month.		Mill Account for the Month.			Face of the Ledger 1st day of February, 1874.	
	Dr.	Cr.	Dr.	Cr.	Total Expense.	Expense per Ton.	Total Credit.	Dr.	Cr.
	dols.	dols.	dols.	dols.	dols.	dols.	dols.	dols.	dols.
Wood cords	1,208.75	..	1,208.75	..	1,208.75	.28			
Quicksilver lbs.	21,422.00	13,494.40	7,927.60	..	7,927.60	1.85	..	13,494.40	
Casting lbs.	10,070.86	4,629.94	5,440.92	..	5,440.92	1.27	..	4,629.94	
Sulph. copper lbs.	926.62	169.62	757.00	..	757.00	..	..	169.62	
Lard oil gals.	50.00	..	50.00	..	50.00	.20			
Coal oil gals.	48.00	..	48.00	..	48.00				
Miscellaneous ..	2,098.90	660.47	1,438.43	..	1,438.43	.34	..	600.00	
Hauling and weighing	8,741.88	..	8,741.88	..	8,741.88	2.04			
Stock	..	20,333.29	..	20,333.29	..	..	..	..	20,333.29
Nevada Mill Co. ..	51,483.29	30,356.72	21,126.57	..	..	..	..	21,126.57	
Mill	..	51,422.82	..	51,422.82	..	..	51,422.82		
Inventory	18,893.96	..	18,893.96						
Labour	6,123.00	..	6,123.00	..	6,123.00	1.43			
Profit and loss ..	..	..	..	..	..	..	..	..	19,687.24
	121,067.26	121,067.26	71,756.11	71,756.11	31,735.58				
		Net credit	..	..	19,687.24				
					51,422.82	7.41	51,422.82	40,020.53	40,020.53

REMARKS.

BRUNSWICK MILL.—MEMORANDUM OF SILVER BULLION DEPOSITED AT THE MINT OF THE UNITED STATES.

No. 513. *At Carson, Nevada, the* 13*th day of November,* 1874 ; *by W. F. and Co. for C. P. G. and S. M. Co.*

Description of Bullion.	Weight.				Fineness.	Value of the Silver.		Gold Parted from the Silver.			Deductions.		Net Value.	
This memorandum is for the information of the depositor, and is of no other value.	Before Melting.		After Melting.					Fineness.	Value.		For Parting, Coinage, and Fine Bars.			
	oz.	dec.	oz.	dec.	1000ths.	dols.	cts.	1000ths	dols.	cts.	dols.	cts.	dols.	cts.
Amalgam.			2,081	80	946	2,483	78	050	2,100	04				
	10.497	60	1,804	00	945	2,302	95	051½	1,920	52				
	10.490	00	1,803	90	947	2,207	50	051	1,901	79				
			2,481	10	945	3,029	79	051	2,615	72			12,617	36
	short 7	60*	2,200	90	947	2,693	34	050½	2,297	56			10,831	63
			10,321	70		12,617	36		10.835	63			23,448	99
								Loss in melting $1\frac{88}{100}$ per cent.						

For Superintendent.

* This difference of $7\frac{6}{10}$ oz. is of little consequence, as this discrepancy occurs between mill and mint scales.

The total expenses of mining, milling, concentration, and furnace work in the Tombstone Mills for 1882-3 is given below:

MINING (21,903 TONS).

	Total.	Per Ton.
Labour	$190,531.00	$8.70
Contract	4,372.10	0.20
Supplies	34,703.48	1.58
Repairs and renewals	980.60	0.05
Ore hauling (21,064.85 tons)	63,204.65	3.00
Total	$293,791.83	$13.53

MILLING (21,474 TONS).

	Total.	Per Ton.
Labour	$56,367.84	$2.62
Supplies	47,152.97	2.19
Repairs and renewals	2,506.08	.12
Total	$106,026.89	$4.93

CONCENTRATION.

	Total.
Labour	9,672.40
Supplies	873.13
Total	$10,545.53

FURNACE.	Total.	Per Ton.
Labour	$7,529.60	
Supplies	23,353.63	
Manganese from the Lucky Cuss Mine ...	9,665.56	
Outside mining	379.62	
Ore purchased	3,752.13	
,, hauling	3,640.18	
Bullion hauling	263.89	
Total	$48,584.61	
SUNDRIES.		
General expense	$8,367.51	$0.38
Administration	22,262.84	1.02
Total	$30,630.35	$1.40
Grand total of mining, milling, and sundries	$430,449.07	$19.86

EXTRAORDINARY.			
Legal expenses	$15,827.49		
Mine construction	8,847.70		
Mill ,, ...	2,188.15		
Furnace construction, including concentrators ...	13,524.98		
Tribute ore	7,559.16		
Adjoining mining claims ...	6,400·00	$54,347.48	$1.90
Total of all expenses ...		$543,926.60	

The details of the work of the mill for the year 1883-4 are given below:

Tons ore received at mill		15,992$\frac{2}{3}$
,, milled		16,042$\frac{2}{3}$
Containing ounces silver		563,282.60
Extracted by amalgamation		402,063.89
Containing ounces gold		4,063.22
Extracted by amalgamation		1,896.27
Average battery assay, ounces ...	35.11 silver	0.253 gold
,, tailing ,, ,, ...	9.81 ,,	0.123 ,,
,, percentage saved ...	72.06 ,,	51.06 ,,

	Total.	Per ton ore.	Per cent.
Pounds (avoir.) amalgam produced	183,307		
,, ,, retort metal ,,	34,070		
Retort metal ratio to amalgam ...	1 : 5.52		
Troy ounces retort metal	496,739		
,, bars	468,666		
Loss in melting	28,073		5.65
Silver in bars, troy ounces ...	394,390		
,, by assay, ,, ...	405,898.57		
Errors of sampling and assay ...	11,508.57	0.72	

	Total.	Per ton ore.	Per cent.
Gold in bars, troy ounces	1,941.11		
,, by assay ,, ...	2,061.44		
Errors of sampling and assay ...	120.33	0.008	
Average fineness of amalgam (both metals)			0.148
Average fineness of retort (both metals)			0.797
Average fineness of bars (both metals)			0.845
Silver remaining in tailings ...	155,382		
Gold ,, ,, ...	2,002		

Salt and blue stone were used, but on account of the falling off in percentage extraction, the quantity of these chemicals has been increased and longer grinding in the pan resorted to. These measures cannot be applied recklessly without amalgamating some of the lead, which, during the year, formed about 8 per cent. of the ore.

The quantities of material used were as follows:

	Total.	Per Ton.
Quicksilver, pounds	20,183	1.258
Salt, pounds	83,850	5.226
Bluestone, pounds...	19,339	1.205
Castings, pounds	59,632	3.717
Wood, cords	2,096¼	0.131
Labour, days	9,453	0.589

The appearance of telluride and lead ores lessened the milling quality of the ore. During the first seven months the ore had an average value of 34.83 oz., of which 7.91 oz. remained in the tails. During the last five the ore contained 35.76 oz., 12.50 oz. remained in the tails. The subsequent treatment of the tails in the furnace caused a portion of this loss to be regained.

The Tables below give one of the curious results of working in the district in which the Tombstone Mill is situated. Taking the battery assays for 1882-3 as the standard of value, and deducting the assay of the tailings, the bullion product is invariably a *plus* one, or in excess of the apparent extraction of the metal, as shown by the assays, and is more marked with high than with low grade ore. The error must of necessity be in the too low assays of the battery pulp, or in the too high assay of the tails. It is probably the latter, but it has been found as yet impossible to definitely fix it.

Tons of ore received at mills ...	... ...		21,186
" milled	... ...		21,474
Average battery assay, ounces ...	32.19 silver		0.18 gold
" tailings "	8.01 "		0.10 "
" per cent. saved	76.07 "		44.44 "
	Total.	Per ton ore.	Per cent.
Pounds (avoir.) amalgam produced	266,026	12.39	
" " retort metal ...	46,888	2.18	
Retort metal ratio to amalgam ...	1 : 5.67		
Troy ounces retort metal	682,637	31.79	
" " bars	641,978	29.89	
Loss in melting	40,659	1.90	5.95
Average fineness of bars silver ...	823.9		82.39
" " " gold ...	2.8		0.28
Troy ounces in bars silver ...	528,946.02	24.63	
" " by assay " ...	519,010.00	24.18	
Gain over assay	9,936.02	0.45	1.91
Troy ounces in bars gold	1,821.99	0.084	
" " by assay "	1,444.52	0.067	
Gain over assay gold	377.47	0.017	26.13

The *plus* in gold is still more striking, if the percentage is considered. The gold assays are made once a month from the collected buttons of the daily assays, and as the latter are made upon samples, taken every half-hour throughout the year, there is no reason why the samples should not be true averages, especially as all questions connected with the mode of sampling have been carefully considered. There is one source of error which may explain the small *plus* of silver. The sample is taken from the stream of pulp at the end of the battery launder, and as the inclination of this trough is somewhat insufficient, an entirely free flow of the pulp is not secured and the heaviest portion of the ore settles in it, removing a certain part of the richest ore from the sample. This is undoubtedly the source of the error in silver returns, but it cannot explain the remarkable error in the gold, unless there is free gold in the ore, heavy enough to settle almost completely in the trough, leaving only the mineralised gold to fall into the sample dipper. Free gold may exist in the ore, but it is not coarse enough to be visible, and it is difficult to see how fine scales should settle so readily.

The mint returns of the Tombstone Mills by months are shown in the following Table, March being estimated. At the mint the bars are remelted, and suffer a certain loss in weight. The

official assays also differ from the mill assays. The differences in results on the year's work are given below.

	Mill Assays.	Mint Returns.	Difference.	
	oz.	oz.	oz.	Per Cent.
Crude bullion ...	641,978	637,008.85	— 4,969	— 0.77
Fine silver ...	528,946	532,372.03	+ 3,426	+ 0.64
,, gold ...	1,821.99	1,813.00	— 8.99	— 0.49

The bank accounts of the mill show the following gains and losses in selling bullion, March not being included :

Shortage on bars, $1,989.85 Gain, $267.87 Nett shortage, $1,721.98.

BULLION PRODUCT.

April 1st, 1882, *to March 31st*, 1883.

(By Mint Returns.)

	Number of Bars.	Gross Weight.	Silver.	Gold.	Base Metal.
		oz.	oz.	oz.	oz.
April, 1882 ...	40	112,864.30	75,258.24	257.58	37,348.48
May	30	76,017.30	35,631.71	102.55	40,283.04
June	18	49,140.70	47,649.62	42.39	1,448.69
July	20	54,020.90	51,154.01	50.08	2,816.81
August... ...	15	42,021.90	41,146.04	50.64	855.22
September ...	18	49,357.35	48,755.54	97.04	474.77
First six months	141	383,422.45	299,595.16	600.28	83,227.01
October... ...	16	44,983.10	43,551.95	163.42	1,267.73
November ...	16	43,032.00	38,297.24	202.00	4,532.76
December ...	14	39,717.50	35,364.61	157.83	4,195.06
January, 1883...	12	34,449.10	30,072.01	169.45	4,207.64
February ...	14	38,845.70	36,375.10	219.55	2,251.05
March	19	52,559.00	49,115.96	300.47	3,142.57
	91	253,586.40	232,776.87	1212.72	19,596.81
Total for year ...	232	637,008.85	532,372.03	1813.00	102,823.82

Average fineness, silver	835.74
,, ,, gold	2.85
Quantity of base metal	161.41
	1000.00

In the assay office 10,100 assays were made, at a total cost of $2864.62 for labour and supplies, or an average cost per assay of 28½ cents.

The cost per ton of ore at the Charleston Mills, Arizona, is as follows :*

* Trans. Am. Inst. Min. Eng., vol. xi., p. 106.

Fuel	$1.05
Chemicals, including quicksilver	0.77
Lubrication	0.04
Illumination	0.03
Castings	0.33
Supplies	0.16
Labour	2.52
	$4.90

COST OF LABOUR.

Crushing	$0.52
Amalgamation	0.67
Power, pumps, &c.	0.47
Foreman, &c.	0.87
Tailings pit	0.11
	$2.64

The cost per ton of ore at the Harshaw Mill in 1881 was as follows :*

Labour	$1.23
Supplies	1.82
Assaying	0.07
	$3.12

COST OF LABOUR PER TON.

Crushing	$0.26
Amalgamation	0.20
Power, pumps and repairs	0.40
Foreman, melter, &c.	0.37
	$1.23

COST OF MATERIALS PER TON OF ORE.

Quicksilver	$0.42
Chemicals	0.07
Castings	0.29
Illumination and lubrication	0.07
Fuel	0.78
Supplies	0.19
	$1.82

Below the actual running† expenses of working Ontario ore, estimated from a production of 50 tons per day, are given:

* Trans. Am. Inst. Min. Eng., vol. xi., p. 100.
† *Ibid.*, vol. viii., p. 557.

No. of Men.	Occupation.	Per Day.	Per Ton
1	Foreman	$10.00	20 cts.
1	Assayer	6.00	12 „
3	Machinists, $4 ...	12.00	24 „
2	Carpenters, $4 ...	8.00	16 „
2	Blacksmiths, $4 ...	8.00	16 „
2	Engineers, $4 ...	8.00	16 „
2	Foremen, $3½ ...	7.00	14 „
9	Dry floor, $3½ ...	31.50	63 „
3	Battery, $4 ...	12.00	24 „
6	Roasters, $4 ...	24.00	48 „
12	Cooling floor, $4 ...	48.00	96 „
4	Carmen, $4 ...	16.00	32 „
4	Amalgamators, $4½	18.00	36 „
1	Retorter	4.00	08 „
1	Melter	4.00	08 „
4	Labourers, $2½ ...	10.00	20 „
4	Watchmen, $3 ...	12.00	24 „
2	Ore floor, $3½ ...	7.00	14 „
3	Clerks, $4... ...	12.00	24 „
		$257.50	$5.15

SUPPLY.

	Per Day.	Per Ton.
Salt, 10 tons, $8	$80.00	$1.60
Quicksilver, 175 lb., 50 cts. ...	87.50	1.75
Wood, 15 cords, $4.50	67.50	3.53
Coal, 12 tons, $8.25	99.00	
Castings	... ...	1.50
Oil and waste	... ...	.25
Sundries, chemicals, &c.	... ...	.50
Hauling from mine	... ...	.49
Charcoal, assaying, and melting	... ...	.25
		$9.87

Total cost, $15.02.

Cost* per ton at three mines is given below:

	Christy Mining and Milling Co.	Stormont Co.		Leeds Co. In 1878.	4 mths., 1879.
Per ton of 2000 lb.,	14,249 tons.		9983 tons.	12,064 tons	4679 tons.
Labour and salaries	$2.85		$2.97	$2.20	
Bluestone, 2.1 lb.	... .31	1¾ lb.	.26	3.22	
Mercury, 1.22 lb.	... .58	1.13 lb.	.57		
Salt, 25.8 lb. ...	... .51	20 lb.	.29½		
Fuel	... 1.31		.45½		
General supplies	... .87		.45		
Incidentals ...	... .41		.12		
Hauling ...	... .73		2.00	.32	
	$7.57		$7.12	$5.74	$4.37

* Trans. Am. Inst. Min. Eng., vol. viii., p. 558.

The operations of the Ontario Mill during the year 1880 are given below*:

		$
Amount paid on labour account	... ...	127,404.81
,, ,, for mill supplies	... ...	151,537.78
,, ,, repairs and construction	...	65,580.00
Gross ore worked in year	tons	13,858
Net ore worked in year (estimated) ...	,,	11,481
Total number of days in operation	... ...	283
,, bullion product in year ...	... ...	$1,344,723.73
,, discount on bullion	... ...	171,241.60

During the company's fiscal year, from February 1, 1879, to February 1, 1880, 15,372 gross or 12,342¾ net tons were treated. The average assay value of the ore was $130.94, and the average tailings assay was $17.49. The percentage extracted was 88. The bullion product for that period was $1,425,003. At the time of examination, however, battery samples for ten days averaged $149.65, and tailings samples for the same time $11.12.

The cost of milling ore is $11.69, or $15 including hauling, superintendence, &c. Supplies cost: Coal, $8.25; and salt, $8 per ton; wood, $4 per cord.

The following Table gives the kind of labour and the wages paid on the Comstock:†

Class.	Wages per shift.	Class.	Wages per shift.
Agitator men	$3.50	Labourers	$3.00 to 4.00
Amalgamators	4.00	Masons	6.00
Blacksmiths	5.00	Oilers	3.50 to 4.00
Blanket sweepers	$3.00 to 3.50	Panmen	4.50 to 5.00
Carpenters	5.00 to 6.00	Retorters	3.50 to 4.00
Chargers	3.50 to 4.00	Refiners	4.00 to 4.50
Driers	4.00	Tankmen	3.50 to 4.00
Engineers	5.00 to 7.00	Teamsters	3.50
Feeders	3.00 to 4.00	Watchmen	3.50 to 4.00
Firemen	4.00	Woodmen	3.50
Foremen	5.00 to 6.00		

A 20-stamp wet-crushing silver mill running twenty-four hours per day generally needs the following crew of men:

2 rock-breaker men.
2 battery feeders.
2 amalgamators.
2 amalgamators' helpers.
2 engineers and in some cases 2 firemen.
1 roustabout.
1 foreman and assayer.

A somewhat larger mill would only require an increase of a

* Reports of the U.S. Census of 1880, vol. xiii., p. 279.

† *Ibid.*, p. 244.

hand or two, and a smaller mill, even a 5-stamp one, could not run with a much smaller force. The usual crew for a 20-stamp silver mill is sixteen to thirty men. Some of the more recently constructed dry-crushing mills, which have more automatic appliances, such as continuous rotary dryers in place of kilns, dispense with a few hands.

The Table on page 437 gives the force employed in three of the small mills.* The Table on page 438 gives the labour and the wages paid at the Ontario 40-stamp dry-crushing mill.† The wages for mill employés range from $2 to $5 for twelve-hour shifts, and seem to be in proportion to the wages paid miners in the district where the works are situated. As a rule, mill employés receive about 10 per cent. more than miners, the difference in the length of their shifts, which for mill hands is almost invariably twelve hours, entitling them to more pay. In one or two districts mill hands were paid less than miners however. Chinese are employed to some extent, and receive from $1 to $2. They work generally as labourers. Foremen are paid from $4 to $10 per day. The number of men employed in the Comstock Mills is given on page 439.‡

The amount and cost of labour in the several States is given on page 440.§ An analysis of the results reached in 160 mills shows that for every ton of ore crushed the labour of one man for 3.9 hours, costing $1.34, is required. For the separate establishments the range is very great, depending upon the character of the ore and the treatment, and upon the efficiency of the machinery, while wide variations are found in the averages for the different States and Territories. The most economical process in point of labour, as in other details, is the treatment of free-milling gold ores by large wet-crushing mills; the most expensive, that of reducing base silver ores, which require chloridising roasting as a preliminary to pan amalgamation. Dakota, where the ores worked are exclusively of gold, and California and Oregon, where such ores largely predominate, are therefore the localities where the greatest saving in labour is effected; while Arizona, Nevada, and Montana show a much greater expenditure of labour, as would be expected,

* Reports of the U.S. Census of 1880, vol. xiii., p. 243.
† *Ibid.*, p. 278. ‡ *Ibid.*, p. 244. § *Ibid.*, p. 245.

THE FOLLOWING TABLE GIVES THE FORCE EMPLOYED IN THREE OF THE SMALL MILLS.*

Class.†	The Grand Prize 20-Stamp Dry-Crushing and Chlorurising Mill.					The Independence - Navajo 10-Stamp Dry-Crushing and Chlorurising Mill.					The Lancaster 10-Stamp Dry-Crushing and Raw-Amalgamating Mill.				
	Average Number employed	Length of Shift. Hours.	Wages per Shift.	Total Number of Hours' Work per Day.	Total Wages per Day.	Average Number employed.	Length of Shift. Hours.	Wages per Shift.	Total Number of Hours' Work per Day.	Total Wages per Day.	Average Number employed.	Length of Shift. Hours.	Wages per Shift.	Total Number of Hours' Work per Day.	Total Wages per Day.
Amalgamators	2	12	$5.00	24	$10.00	2	12	$5.00	24	$10.00	2	12	$5.00	24	$10 00
Amalgamators' helpers	2	12	4.00	24	8.00	...	...	...	...	...	2	12	4.00	24	8.00
Chlorurisers	2	12	5.00	24	10.00	2	12	5.00	24	10.00					
Chlorurisers' helpers	2	12	4.00	24	8.00	2	12	4.00	24	8.00					
Battery feeders	2	12	5.00	24	10.00	2	12	4.50	24	9.00	2	12	4.50	24	9.00
Engineers	2	12	5.00	24	10.00	2	12	5.00	24	10.00	2	12	5.00	24	10.00
Firemen	4	12	4 00	48	16.00	2	12	4.00	24	8.00	3	12	4.00	36	12.00
Melter and retorter	1	12	4.00	12	4.00	...	...	...	...	...					
Dry-kiln men	6	12	4.00	72	24.00	4	12	4.00	48	16.00	2	12	4.00	24	8.00
Blacksmith	1	12	5.00	12	5.00	...	...	...	...	...					
Labourers	4	12	4.00	48	16.00	...	...	...	...	...	1	12	4.00	12	4.00
Sage-brush wheelers	...	...	...	...	......	...	...	...	...	...	2	12	4.00	24	8.00
Total	28	...	...	336	$121.00	16	...	...	192	$71.00	16	...	...	192	$69.00

* Reports of the U.S. Census of 1880, vol. xiii., p. 243.

† The assayer and foreman are included in staff.

in view of the character of their ores. In Colorado and Idaho the different classes of mills are nearly balanced, and the labour employed in crushing a ton of ore closely approximates the average for the whole country. It should be observed that while in tonnage the gold mills take the lead, the silver mills treat ore of a higher grade, so that a comparison based upon the bullion product would show somewhat different results.

CLASS.	Number employed.	Length of Shift Hours.	Wages per Shift.
Foreman ...	1		
Chief engineer ...	1		
Assayer ...	1		
Clerk ...	1		
Night boss ...	1	12	$4.50
Ore weigher ...	1	10	4.00
Rock-breaker ...	1	10	3.00
Carmen and drying-furnace feeders ...	2	12	4.00
Ore driers ...	12	8	3.00 and 3.50
Battery feeders ...	3	8	4.00
Amalgamators ...	4	12	4.50
Carmen ...	2	12	4.00
Furnace men ...	6	8	4.00
Cooling-floor men ...	12	8	4.00
Engineers ...	2	12	4.00
Firemen ...	2	12	3.50
Salt feeders ...	2	12	4.00
Watchmen ...	2	12	3.00
Carpenters ...	3	10	3.00 to 4.00
Machinists ...	2	10	4.00
Machinists' helpers ...	2	10	3.00
Retorter ...	1	10	4.00
Melter ...	1	10	3.50
Storehouse keeper ...	1	10	3.75
Blacksmiths ...	2	10	3.25 and 5.00
Wood haulers and team ...	2	10	7.50
Assayer's helper ...	1	10	2.50
Tailings-pit man ...	1	10	3.00

To produce a dollar in gold bullion costs 8 cents for mill labour in Dakota; in Nevada, though over five times as much work is required per ton treated, the cost for mill labour per dollar of silver bullion produced is 7 cents, or practically the same proportion. The Table* on page 439 shows the average number of hours' work and cost of labour per ton of ore crushed in the principal mining States and Territories, and is based on the treatment of nearly a million and a half of tons.

* Reports of the U.S. Census of 1880, vol. xiii., p, 245.

Mill.	Number of Men employed.	Amalgamators.	Acid Makers.	Battery Feeders.	Blacksmiths.	Blanket Sweepers.	Bluestone Maker.	Bullion Refiners.	Carpenters.	Cooks and Waiters.	Engineers.	Firemen.	Foremen.	Labourers.	Lead Burner.	Machinists.	Metal Roaster.	Millwright.	Oilers.	Retorters.	Tank-men.	Teamsters.	Watchmen.
Brunswick . . .	40	4	...	...	2	4	...	...	1	...	...	...	...	9	...	...	...	...	2	2	14	1	1
California . . .	72	7	...	...	1	8	...	...	1	...	4	3	3	33	...	2	...	...	2	...	6	...	2
Excelsior . . .	12	...	...	...	...	...	...	...	...	1	2	...	1	8									
Franklin . . .	15	2	...	...	2	2	...	...	1	...	...	...	...	8									
Lyon	46	...	2	...	1	...	1	...	...	3	...	3	1	23	1	...	5	...	...	...	...	6	
Mariposa . . .	18	3	...	...	...	...	...	...	...	...	3	...	...	7	...	1	...	...	...	...	4		
Morgan. . . .	33	2	...	...	2	1	...	...	1	...	3	...	...	9	...	...	...	...	3	1	10	1	
Omega.. . . .	12	...	...	...	...	...	...	3	...	...	...	...	...	7	...	...	...	...	...	...	...	...	2
Scorpion G. and S. M. Co.*	26	4	...	3	...	2	...	...	...	1	5	...	...	7	...	...	...	1	...	...	2	1	
Trench. . . .	26	3	...	...	1	...	...	...	...	...	3	...	...	4	...	...	...	...	...	...	14	...	1
Woodworth . . .	31	2	...	...	1	...	...	...	1	...	2	...	...	25									
Total . .	331†	27	2	3	10	17	1	3	5	5	22	6	5	140	1	3	5	1	7	3	50	9	6

* Includes Boston and Douglas Mills.

† In addition to the number of men here classified there were employed at other mills, which were either idle or running but a short time, 66 men, as watchmen, labourers, &c. The total amount paid as wages during the year 1880 by the Comstock Mills was $372,767.19.

State or Territory.	Number of Mills.	Tons Treated.	Total Number of Men employed.	Foremen.	Amalgamators.	Day Labourers.	Other Workmen.	Total Number of Hours Work done during the Year.	Total Sum paid on Labour Account, exclusive of Staff.	Average Number of Hours per Ton crushed.	Average Amount Paid for Labour per Ton Crushed.
Arizona . .	11	15,946.40	137	3	23	64	47	200,340	$71,717.00	12.563	$4.497
California .	33	419,883.50	301	13	76	147	65	1,078,742	291,577.00	2.569	0.694
Colorado . .	28	140,117.35	197	23	22	57	95	579,574	146,024.00	4.137	1.042
Dakota . .	22	506,238.00	259	13	35	147	64	825,001	251,759.00	1.629	0.497
Idaho . . .	7	16,391.25	59	1	8	9	41	70,128	22,017.00	4.278	1.343
Montana . .	13	47,801.00	144	9	17	62	56	373,746	191,609.00	7.819	4.008
Nevada . .	33	237,604.75	742	34	96	147	465	2,063,813	739,149.00	8.686	3.111
Oregon . .	5	12,437.00	27	2	6	14	5	41,684	10,250.00	3.352	0.824
Utah . .	8	71,059.50	180	3	19	10	148	512,757	249,804.00	7.216	3.515
Total . .	160	1,467,478.75	2046	101	302	657	986	5,745,785	$1,973,906.00	3.915	$ 1.345

The fuel used is generally wood.* It is of many different kinds and qualities. In some mills in Nevada sage-brush is used under the boilers. The leading kinds of wood are: Mountain mahogany, weighing from 3200 lb.† to 4400 lb.‡ per cord; mesquit, weighing from 2500 lb. to 3500 lb.; nut-pine, with very variable weight depending on seasoning, size, and shape of sticks, &c., may be calculated at between 2000 lb. and 3300 lb.; sugar-pine, from 2000 lb. to 2300 lb.; cedar, 1800 lb. to 2500 lb.; cottonwood, 1500 lb. to 2300 lb. As near as can be determined 156 lb. of sage-brush are equal to 100 lb. of good cedar. The averaging of different weights of the same kind of wood is impossible, as the weight of a given kind of wood depends upon the locality of its growth, which affects its grain, the seasoning, and the closeness with which it is corded.

It is almost impossible, on account of the different character of the wood burned and the differences of method and machinery, to determine the quantity of fuel consumed under the boiler in working a ton of ore. For wet-crushing silver mills three-quarters of a cord per stamp is probably the maximum, in which case the wood is of very poor quality or the machinery very inefficient; the average is thought to be less than half a cord. In roasting-mills, which are always dry-crushing, about one-third of a cord is consumed to the stamp. This does not include the wood necessary for roasting.

The power which drives the mills is usually steam. Sometimes water alone is used, as in the Eureka and Brunswick Mills, and sometimes both water and steam. When water is used at all, it is always best to have an engine in reserve to run the mill in case of freshet or drought. The Brunswick Mill has two La Fille wheels, of 250 horse-power each. The driving wheel is 9 ft. in diameter. The driving belt is 42 in. wide. In July the water here gives out, and the wheels are set so low that in the spring freshets the water sometimes backed up against the wheels, and prevented their working. The question of the use of steam or water power is one which has to be settled every time a new mill

* Reports of the U.S. Census of 1880, vol. xiii., p. 247.

† This weight is from data of the Eureka and Palisade Railroad.

‡ This weight is from data of the Richmond Mining Company.

is built. The opinion of most engineers would at once be in favour of steam. As it has to be used in the mill the year through, for heating the pulp, a part of the equipment of a steam engine must necessarily be in the mill, and in the end it would in most cases be cheaper to use steam power. The loss occasioned by a few weeks' stop of a large mill would very soon pay the expense of the engine and boilers, and it therefore seems poor economy to rely on water alone. Water in all new mining countries is an exceedingly uncertain reliance, and a mill depending on it alone will generally be idle from a quarter to half the year. Where water power is the only force used, the saving over steam is 80 per cent., since some steam has to be used; but this is true only on the supposition that there are no stoppages.

The total amount of water and power required in a large mill is about as follows.* Each stamp generally uses 10 lb. of water per minute, each pan 16 lb., and each settler 9 lb. In cases where water is very scarce, the whole water of the mill may be run into a reservoir which thus becomes a tailing reservoir, and is pumped back with a loss of 20 per cent. In such a case there would be required for the stamp, 2 lb.; for the pan, 3.2 lb.; and for the settler, 1.8 lb.

The total amount of water required for a 60-stamp mill will be:

	lb.
244 horse-power requires, per minute	183
60 stamps	600
28 pans	448
14 settlers	126
Total amount of water required per minute	1357

Of this amount 1174 lb. can be used again, at a loss of 20 per cent., and the 183 lb. for the engine can be condensed at a loss of 50 per cent.:

	lb.
20 per cent. of 1174 lb.	234.8
50 ,, 183 ,, ...	91.5
	326.3

This is the total amount of water consumed when a reservoir is used.

* For this information I am indebted to Mr. J. M. Scott, of San Francisco.

Stated in gallons the quantity of water used is: For the boiler, 7½ gallons per horse-power per hour. For each stamp, 72 gallons per hour. For each pan, 120 gallons per hour. For each settler, 60 gallons per hour.

If the water used in the battery, pans, and settlers is collected in settling tanks it can be re-used with a loss of about 25 per cent. In general the amount of water may be stated at from 2000 to 2500 gallons per ton of ore treated.

These details are of much less importance than they formerly were when there were no ditch companies. They are intimately connected with the question of saving slimes and tailings, and are of vital importance in all countries where water is scarce.

The following Table gives the horse-power required for four different mills.

—	1. 10-Stamp Wet Crushing Mill.		2. 20-Stamp Wet Crushing Mill.		3. 60-Stamp Wet Crushing Mill.		4. 10-Stamp Dry Crushing Mill.	
	No.	Horse-power required.	No.	Horse-power required.	No.	Horse-power required.	No.	Horse-power required.
Blake's crusher No. 2	1	6	1	6	1	5.5	1	6
Ore feeders . .	2	0	4	0	...	...	2	0
Stamps of 750 lb. with 90 drops .	10	12	20	23	60	67.5	10	12
Rotary furnace 40in. in diameter . .	...	...	...	...	...	...	1	4
Pans 5 ft. in diameter . . .	6	30	12	60	28	112	4	8
Settlers 8 ft. in diameter . . .	3	9	6	18	14	28	2	8
Concentrators .	...	...	...	...	3	6		
Friction . . .	...	7	...	13	...	25	...	9
Total horse-power	...	64		120		244		45

1. Treats 18 to 20 tons in 24 hours. 2. Treats 40 tons a day of 24 hours. 3. Treats 130 to 140 tons a day of 24 hours. 4. Treats 15 to 18 tons a day of 24 hours.

The engines of all the 60-stamp mills of Nevada are made at least double the calculated power. The pressure of steam ordinarily used in the boilers is about 80 lb. In a few mills it is as

high as 90 lb, and is sometimes even carried up as high as 100 lb. It is rarely less than 70 lb.

The capacity of the mills depends upon the number of stamps, the hardness of the rock, and the number of days in the year which the mill can run, which is regulated chiefly by the supply of ore, but sometimes by variation in the water supply. A few mills treat as high as 375 tons to 400 tons in twenty-four hours. Counted by stamp capacity, it varies with the hardness of the rock from 1½ tons to 4 tons per stamp in twenty-four hours. The great majority of the large mills are owned and operated by companies who own the mines. On the Comstock many of the mills are custom mills, but it is generally true that the large custom mills were originally built by mining companies, and have fallen into the hands of private parties, either on account of the failure of the mines or want of proper management of the mills.

The cost of a small mill of less than fifteen to twenty stamps, with all its appliances, will not be less than $1000 per stamp. A 60-stamp mill, like the Consolidated Virginia, Brunswick, and Eureka mills of Nevada, can be built for from $600 to $700 per stamp. Notwithstanding this great outlay, the mills—which are generally owned by companies who contract to work the total product of the mine—almost always declare large dividends while the mines are often not worked to a profit. The way in which the ores are purchased leaves such a margin of gain to the skilful mill manager that there is very little left to pay the miner for his trouble, unless the ore is very rich, or he is in some way interested in the mill. The owners of custom mills who have no mine interest or contract, but are obliged to go into the open market to purchase their ore, cannot therefore compete with the owners of mills who have large mine contracts; and must either sell out to them, or be absorbed by them. Hence, as the mill makes most of the profit, the miner is generally anxious to erect a mill before he has fairly opened his mine. A great many really good enterprises in the West and elsewhere have in this way been ruined, merely because the capital which should have been invested in opening the mine has been spent in surface improvements before there was a mine to justify them. Monuments of the folly of spending the capital of mining com-

panies on surface improvements before the mine is fairly opened exist in the shape of ruined buildings and rusty machinery which belong to no one, and which no one can buy, sell, or use which are going to decay as fast as the elements can destroy them. Many of these ventures were the honest endeavours of inexperienced persons, but they were guided by want of skill and judgment, the small capital available being spent on surface improvements until there was no money left to work the mine. In many of these cases if the funds had been expended on the mine, the profit from the sale of ore would have been sufficient to erect the surface plant, and the investment instead of being a failure would have been successful. A few of the failures were the result of speculations for purposes not strictly honest; but they have all brought mining more or less into discredit as a haphazard investment, when in reality, with the same foresight, prudence, and management, which is given to other commercial enterprises, mining and milling would pay more than double the legitimate profits of ordinary business.

CHAPTER IX.

THE TREATMENT OF SILVER TAILINGS.

In every silver mill there are two streams discharging, one from the battery and one from the settlers. As they are treated differently, different names are given to the residues which accumulate from them. The pulp which runs out of the settlers and agitators is called "tailings" or "sand." The very fine clayey material which comes from the stamps, and is too light to be deposited inside the mill either in the pulp or slime vats, is called "slimes" or "slums." The tails have been in contact with mercury, the slimes or slums have not. There are thus slimes from the slime vats in the mill, which are treated with the pulp and slimes outside of the mill, which if caught are collected in reservoirs. The part of the pulp which has been reduced to a slimy condition in the pan is called pan slime, and hence there are three kinds of slimes: mill slimes, collected in the slime vats of the mill, which are treated in the pans; battery slimes, which are the overflow of the slime vats; and pan slimes, which discharge from the agitators with the tailings, and are only accidentally separated from them in the lower parts of large reservoirs. The battery slimes are usually allowed to escape, or are caught in large reservoirs below those for the tailings. The battery slimes and tailings are treated together in concentrators, such as the Frue Vanner, or on blanket sluices principally, to catch the mercury and amalgam and heavy particles of ore, and the concentrates are treated in pans.

If the ore has been properly worked in the pans, very little gold or silver can be extracted directly from the tailings and pan slimes. They are for the most part in the state of sulphurets, which cannot be separated without thorough oxidation, produced either by roasting, by chemical action, or long exposure to the action of

the weather. They contain, however, a certain portion of mercury and amalgam, and occasionally sulphurets containing gold which will sometimes pay to concentrate and separate.

The battery slimes are sometimes richer, but they are usually poorer than the ore; they generally will assay about 60 per cent. of its value. They contain proportionately much less gold than the ore. The quantity of slime depends upon the amount of water used in the battery, the method of settling, and the quantity of clay in the ore. With hard ores the amount of slimes in the best mills will be 2 to 3 per cent., with soft ores it will be more than double that quantity, and sometimes are as high as 15 per cent. Whatever gold there is in them is very flaky and likely to float; the silver is in a very fine state of division and also likely to float, and is also to some extent in the form of sulphurets. The slimes are worth from $15 to $20, and often more, and can be profitably treated. When water is scarce the slimes are very advantageously treated without any special arrangement being made for it, as all the water from the slime vats may be pumped up with the slimes in it, and made to pass again through the mill. By this method there are no battery slimes. It is well worth consideration whether this method would not pay even when water was plenty. The amount of water to be paid for would be less, and the gain in yield would probably more than pay for the pumping, if properly managed. The pan tailings and slimes could be caught with a sufficient number of tanks, but the mills have not usually been built with reference to it, and could often not be altered, as there is generally not sufficient space around them for the purpose. Frequently the slimes are not valuable enough to justify much expense, so that in most of the mills only a rough attempt is made to catch the mercury and amalgam, and very large quantities of them go to waste.

When the ores are treated raw, the tails are always of value. They are allowed to weather, salt being sometimes added to them to hasten the decomposition. After a time, more or less long, they are retreated. Generally there is sufficient salt remaining in them to effect this and to render the silver subject to attack by the mercury. A certain amount of exposure seems to be necessary,

but even after some time tails rarely yield over 50 per cent. of the assay, and often give much less. The tails from roasted ores are much more difficult to treat and the yield from them is much less. The treatment of the slimes varies but slightly from the treatment of the ore, differing only in these respects, that pans of much larger dimensions are used, and that the slimes do not require grinding. Generally the battery slimes are settled in a series of reservoirs. The richest material will always be in the first one. There were a large number of such slime reservoirs of great extent that have been filled to the depth of 8 ft. to 10 ft. in Dayton Cañon, below Virginia City, Nevada, where a number of mills have been established to treat them; but the best of them do not get more than 60 per cent. of the assay of the slimes, even where they use very large amounts of chemicals, as most of them do. The bullion produced is never fine, and is less so as the amount of copper sulphate is larger. The loss of mercury is so great that some of the tail mills, as mills treating tailings and slimes are called, purchase ore to mix with them. This loss has given rise to the invention of a number of machines for saving it.

The tailings were formerly generally treated in blanket sluices, the attempts to treat them on most of the concentrating machines not having been, until recently, successful. The sluices are troughs 20 in. wide with sides 2 in. or 3 in. high, and sometimes 1700 ft. or 1800 ft. long, with a grade of 6 in. to 10 in. in every 12 ft.; a number are usually placed side by side, generally three. Sometimes as many as fifteen to twenty and even more are arranged together. They are covered with strips of coarse blanket, which are laid on the sluice, and can be easily removed to be washed. When the sluices are long they are tarred on the underside to keep the blanket from rotting, in which case they are nailed to the sluice and swept. When not tarred they are taken up and washed. The tarring is done by drawing the blanket over a bath of hot liquid tar.

As the stream of tailings runs over these blankets the heavier portions are caught; the lighter portions are washed away. The loose blankets are taken up at intervals of about twelve hours, and washed in tanks, the tarred ones are cleaned by a man who

walks over the blankets, brushing the surface lightly with a broom, thus disturbing and distributing the material, and aiding the action of the water. The material caught in the blankets is thus swept into under-sluices beneath the main sluice, and is caught in tanks. While the blankets of one sluice are being cleaned by washing or sweeping, the stream is turned on to the others; so that whatever the number of sluices required to do the work may be, there must always be one more. These sluices are sometimes owned by the mills, but they generally belong to contractors; they usually cost about $1 per foot, including the blankets. The concentrations from both kinds of blankets are worth from $18 to $30 per ton, and are treated in pans. What passes the short blanket sluices is generally accumulated in reservoirs. Below Virginia City there are immense quantities of these tailings, collected in artificially constructed reservoirs, from which when they were full, the water was turned off, the dam and the sides removed, and the material, which soon becomes thoroughly dry, carted to the mills for treatment. It is doubtful whether it will pay to save the residues of long sluices.

As illustrations of the way this kind of work is done, the Eureka Mill and Woodworth sluices have been selected. At the Eureka Mill the discharge from the agitators is run into a large tank from which it is raised by an Archimedean screw 32 ft. long and 16 in. in diameter, which discharges into a head box 10 ft. long, 1 ft. wide, and 16 in. deep. This box has six gates which open on to the same number of sluices built side by side, each of which is 1200 ft. long. They are 20 in. wide, and are separated by strips of ½ in. board, 3 in. high. Five of these sluices are used at a time; the sixth is held in reserve. The blankets are 10 ft. to 12 ft. in length, and are not cut. They are tarred on the underside, and are fastened with five tacks at the upper end, and by a cleat tacked to the sides. The lower end is loose and laps over the blanket below it from 6 in. to 12 in. Every 200 ft. there is an opening across the sluice 3 in. wide, which is covered, during the time the tails discharge over it, with a sheet-iron cover, which communicates with a trough running across the sluice below. This opening is always placed at the head of a blanket, so arranged that the lap of the blanket

covers it while the tails are running. When the blanket is to be swept the cover is removed, the end of the blanket drops over the edge of the opening, and the blanket concentrates are carried into the trough below. The upper end of the sluice is swept every four hours, the lower end once in six hours. Six tons of concentrates are collected in a day. It takes three men per shift of twelve hours to do the work. The concentrates here have the same assay value as the ore, and are treated in the same way with the addition of one third of slimes.

The best example of long sluices is the Woodworth sluices of Dayton, Nevada.* They are the largest which have ever been built, and are designed to treat all the tailings from twenty-five to thirty mills in Gold Cañon, or 262 stamps, being an average of 2 tons a day for each stamp. These are all collected in the sluice at the Bacon French Mill. This sluice is 18 in. square, 3½ miles long, with an average grade of 4 in. to the rod. It is never cleaned, but carries the collected tailings to the Woodworth sluices. Figs. 181 to 183 give perspective views of parts of this sluice. Fig. 181 shows the upper part, Fig. 182 the middle, and Fig. 183 the end of the sluice. It is composed of twelve sluices S side by side, each one of which is 19 in. wide. They are separated by strips of wood 1½ in. wide and 3 in. high, and form together a table 22 ft. wide and 1700 ft. long. These sluices have a grade of 2 in. to the rod, and are supported on trestles 4½ ft. apart. For convenience of working, the group is divided into sections of 150 ft., but each sluice is continuous. At the head of the sluice the tails are discharged from the sluice A, Fig. 181, into a head-box B, made of two compartments included in the same structure. These are each 3 ft. long and 2 ft. deep, and the whole width of the sluices. The bottom of the first one, B, is 20 in. above the bottom of the sluice, and has three openings 3 ft. wide and 4 in. high, which discharge into the distributing box C, which has twelve openings 4 in. by 8 in., one for each sluice. All the openings in C are provided with gates D, so that all or any number of them can be closed at a time. In front of the box B is another one E, of the same size

* These sluices were designed and constructed in 1874 by W. H. Armstrong, for the Dayton Mill, of which he was then superintendent.

and shape, which is fed by clear water from the two lines of galvanised iron pipe F, 5 in. in diameter and 2156 ft. long, on one side of the sluice, and on the other supplies it, through the gate G, to the side box H, which is 8½ ft. wide and 10½ ft. deep, and carries the clear water to the head of the sections; it is consequently 150 ft. shorter than the whole sluice. Here it discharges, through the openings L, into boxes M 6 in. by 6 in., which cross the sluice at right angles, and have, like the clear-water head-box E, a 3 in. round hole I for each sluice, which can be closed with a plug when the blankets are not being washed. These troughs are 1 ft. above the sluice, and are supported on four inclined pieces shaped so as to leave the current free. At the end of each section there is a slot K in the bottom of the sluice, 3 in. wide and 17 in. long, closed by a sheet-iron cover, which opens into an inclined sluice 6 in. wide and 12 in. deep, which runs under the whole width of the table, and at right angles to it, and discharges into a trough O of the same size, which runs parallel to the table, and carries the concentrates to the reservoirs. This slot in the bottom of the sluice is closed when the sluice is working, and the blanket laid over it, as at the Eureka Mill.

At the end of 1200 ft. there are ten settling tanks 9 ft. long, 8 ft. wide, and 4 ft. deep, into which the sweepings are discharged. At the end of 1600 ft. there are two of the same size, and at 1700 ft. two more, making fourteen in all; 140,000 ft. of lumber, 24,000 ft. of blankets, 19,000 ft. of which are in use at a time, 2 tons of nails and spikes, and 1600 gallons of coal tar were used in the construction of the sluice. The total cost of construction was $21,500. It requires to operate it about 627 gallons of water per minute, which is pumped to a height of 12 ft. vertically, and carried 2156 ft. horizontally through a 5 in. iron pipe. The pump is driven by the water-wheel of the Dayton Mill. Each one of the sluices is covered with blankets 20 in. wide by 6 ft. long. They are tacked on the upper end with five 8-oz. tacks, and by one tack every 6 in. on the sides. Before they are put down they are coated with coal tar on the under side. This is done by laying the blanket on the hot tar in a tank made for the purpose, whose bottom slants 35°, so that the top of one of

the sides is but a few inches above it. The blanket is drawn over the surface of the tar from the deep side of the tank and then over this side, the edge of which scrapes off the excess of tar. This tank is 6 ft. long, 30 in. wide, and 12 in. deep. Only the under side is coated in this way, leaving the nap entirely free. The tarred blankets last nine to twelve months, but if not tarred they only last three months. They are taken up every two months to be dried, and are then shaken; at the same time the bottom sluices are carefully cleaned. The blankets work better as they get older.

To work the sluice, the slime from the sluice A is discharged into the tank B, Fig. 181, and from there into the distributing box C. The amount of slime discharged by the sluice is from 60 to 70 cubic inches per minute. It is discharged upon the blankets from the gates D in the trough C, the amount being regulated by opening or closing the gates. It is thus distributed over all the sluices but one, or as many as it is desirable to have working, and is spread over the blankets.

The first three or four sections are constantly swept, so that each section will be swept every six hours. Before commencing to sweep, the supply of tails is cut off, and clean water turned on to the sluice from the clean-water box for half an hour. The opening in the bottom of the sluice at the end of each section, which is kept closed while the sluice is not being swept, is now opened, the cover is removed, and the end of the blanket allowed to drop into the sluice below, where it remains until the men clean up this sluice. The sweeping is done with an ordinary broom, clear water running all the time. The sulphurets caught in the nap of the blankets are thus discharged into the lower sluice. The length of time for sweeping is generally calculated on the upper sluices, so that three to four of them are washed in a little less than three-fourths of an hour. Each one of the sluices is thus swept in eight hours, or three times in the twenty-four. On the lower ones they are swept only once an hour, so that each sluice is swept once in twelve hours. Seventeen men do the whole work of the sluices, twelve of whom are sweepers. They work twelve-hour shifts, and were paid, in 1874, \$3 a day for sweepers, and \$4 for the overseers. Afterward Chinese

FIG. 181. THE WOODWORTH SLUICES AT DAYTON, NEVADA.

labour was employed, so that now the sweepers get only $1.54 a day. The work is arranged so that the sluices produce one ton of concentrated sulphurets for each sweeper. Fully three-fourths

FIG. 182.

FIG. 183.

THE WOODWORTH SLUICES AT DAYTON, NEVADA.

of this comes from the upper sluices. The actual cost of concentrating, with labour at $3, was about $4 per ton of sulphurets, with the cost of labour reduced it was proportionately less.

In any case the cost will be more than covered by the gain in mercury. The assay of the concentrates in October, 1874, just after the sluice commenced to work, was $25.49 In November of the same year it was $28.90. These concentrates are worked in the mill to 70 per cent. of their assay value.

The attempt to work the slimes by direct amalgamation was for a long time unsuccessful, owing probably to the precious metals being in the state of sulphurets, which require to be roasted, or of their being included in pyrites so that they are not reached by the mercury. Roasting is generally too expensive, so that this material is either thrown back into the battery, or worked in the pans. Their quantity has been very much diminished by increasing the number of slime vats in the mill. Since the decay of the Comstock Mines most of the mills on Carson River are working on Comstock tails, which were formerly considered not worth treating, but which the low price of mercury and improved methods have made possible to treat with profit. They are now treated in arrastras and pans, or in pans alone, by largely increasing the amount of chemicals, which are usually only salt and copper sulphate, over that formerly used. The amount of chemicals and mercury must be determined in each case by making assays; for the baseness of the bullion produced increases with the quantity of copper sulphate, and cases have been known where the silver produced has been only .099 fine. Having once determined the average fineness of the bullion for a given quantity, this quantity is the one best suited for the slimes in hand. In general the less the amount of chemicals, the finer the bullion, but the greater the loss of precious metal will be.

As battery slimes require no grinding, it would seem most advantageous to charge the quicksilver directly into the pan, but experience has shown that better results are obtained by charging the chemicals with the slimes, and thoroughly mixing them for about two hours with the muller up, and then adding the quicksilver. Pan slimes sometimes form a pasty mass which might hold the finely divided quicksilver and carry it off. In treating such material one-half its bulk in "sand," as the tails from the pans are often called, and sometimes more, is added to it. When

the quantity added is large the prepared charge sometimes assays only $6 to the ton. With slimes assaying from $20 to $30 a ton, the average quantity of copper sulphate for tailings is from 10 lb. to 12 lb., and sometimes as high as 20 lb., most of which is regained in the refining, and of salt from 20 lb. to 25 lb. to the ton of pulp. The charge is worked for four hours, or six hours in all, and then drawn into the settler. About 60 per cent. of the assay value is obtained. The yield of the pan being from 60 to 75 per cent., if the slimes are successfully treated, may bring the yield of the ore up to 85 or 90 per cent. Tails are often treated in the same way, but undergo a preliminary grinding in an arrastra.

In treating slimes the greatest source of expense is the apparently inevitable loss of mercury. With all the best appliances it will generally be as much as 4 lb. to the ton of slimes. In the treatment of ore it will not generally exceed from 1 lb. to 1½ lb., and sometimes averages as low as ¾ lb. This loss is probably owing to the slime forming a coating over the globules of mercury, and preventing their uniting. When ore is treated with the slimes the loss in mercury is always found to be less, and the yield of silver higher; this is probably owing to the more or less grinding action of the ore which cleans the mercury. It was supposed for a long time that sodium amalgam, potassium cyanide, and a number of other chemicals, would have the effect of preventing the loss of mercury to a very considerable extent, and they undoubtedly do temporarily "enliven" the quicksilver. But their action is only temporary, and it is doubtful whether it is of any great amount of use. As these slimes, and sometimes low grades of ore, are poor, it is necessary to work them in very large quantities in order to make them pay. This is done by using very large pans, and it is mostly for this purpose that the largest pans have been invented.

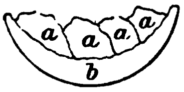
The examples of the treatment of tailings, slimes, and concentrates from the blanket sluices in pans, are taken from the Woodworth and Lyon Mills in Dayton, Nevada. The Woodworth Mill treats the slimes mixed with ore worth $20 a ton, and for that purpose has six batteries of four stamps each. It has twelve Horn pans, six settlers, two agitators, and one clean-up pan.

The Lyon Mill treats only tailings, slimes, and the concentrates from blanket washing. It has ten Wheeler pans, with wooden sides and covers, five settlers, and one clean-up pan. The slimes have been deposited in reservoirs from the mills above. The dam and sides have been removed, and the material, which is perfectly dry, dug out of the bank, and carted to the mill. They were sometimes worth from \$30 to \$40; their general averages were, however, from \$15 to \$20. As they have not passed the pans they are much richer than the "sand" or tails. Since the Comstock Mines have ceased to work, tails as low as \$3 to \$4 are treated in charges of 2 tons to 4 tons. During the last years of prosperity they were working, they averaged from \$5 to \$7. The concentrates come from the Woodworth sluices. In the Woodworth Mill the pans are 5 ft. inside diameter, have eight dies and six shoes which are 18 in. from side to side on the widest part. One end of the shoe is only 4 in. wide, the other is 18 in. The ends are joined with a curve, so as to form a decided point on the long side of the shoe. The diameter of the shoes and muller is 4 ft. 10 in., so that there is 1 in. between them and the side of the pan. The charge for the pan is 1600 lb. of tailings and 400 lb. of pulp; but when of ore alone, the charge is 3000 lb. The muller revolves eighty-five times a minute; 20 lb. of sulphate of copper and 40 lb. of salt are put in with the charge. When slimes are treated the muller is never down, and 200 lb. of mercury are introduced at the end of two hours. The whole operation lasts 4½ hours. When ore is charged the pan grinds for three hours, then 100 lb. of mercury are added, and the operation continued with the muller up for 1½ hours. When the operation is finished the contents of the pan are drawn into settlers. The sides of the pan are made of sheet iron and lined with wood. This is done because it has been found that the mercury forces itself through the pores of the wood, and also that there is a loss of mercury where it leaks. This loss is prevented by the sheet-iron sheath, or if it takes place, the mercury is caught in the groove round the bottom of the pan. In the Lyon Mill the pan is 5 ft. in inside diameter and 4½ ft. deep. The charge is 3 tons of dry tailings, which are mixed with water, discharged into the pan through a 1½ in. pipe. During

the time the charge is being made the muller revolves seventy times per minute. The pan is filled to within 6 in. of the top, or just as full as it can hold without slopping, the pulp being in a state of thick mud. The shoes are always raised $\frac{3}{4}$ in. from the bottom. Half an hour after the charge is introduced, 40 lb. of salt and 74 lb. of acid copper sulphate, containing a large amount of iron sulphate, is added. This material is the refuse from the crystallisation vats of the "blue stone" manufactory owned by the mill. It could not be sold, and the company find it cheaper to use it in their own mill than to re-crystallise it. The best results were always obtained when the bullion was made designedly very base through the addition of a great excess of the impure copper sulphate, the fineness of the bullion was therefore purposely kept down to from 150 to 250. One hour after, the charge of 100 lb. of mercury is introduced, the full amount being charged at one time. In twenty-nine days 340 flasks of $76\frac{1}{2}$ lb. each were used on nine pans. The pan revolves for three hours after the mercury is added, and is then drawn into the settlers. In the Woodworth Mill the settlers are 10 ft. in diameter, 3 ft. deep, and the arms make fourteen revolutions per minute. In the Lyon Mill they are 8 ft. in diameter, 4 ft. 8 in. deep, and making fifteen revolutions a minute. No mercury is ever taken from the pan; it is all discharged into the settler from a pipe in the bottom. Water from a 2-in. pipe runs into the pan while it is discharging, until the settler is filled to within 6 in. of the top. The settlers are so large that sufficient water can be run into them to cause the settling of most of the float mercury and amalgam. Each pan has its own settler which is kept in motion during the whole time a charge is in the pan. This will generally be about three hours or a little more or less, depending on whether the pan takes three or four hours to work. It is emptied three quarters of an hour before the pan is ready to discharge. From the bowl on the side of the settler the mercury is dipped out to go to the strainers, which are surrounded by a sheet-iron sheath, reaching nearly to the bottom of a large iron vessel. After straining, the amalgam is weighed. It contains six parts of mercury to one of silver. It is retorted in the usual way, and is worth from $0.50 to $0.60.

The loss in mercury is only $\frac{1}{7}$ lb. to the ton; but this exceedingly small loss is owing to the fact that when blanket concentrates are treated there is a gain in mercury from them which reduces the apparent loss. In one of the mills of this vicinity, where twenty pans were run on tailings alone, nine flasks of mercury were gained over and above all the losses in six weeks. The real loss is, therefore, much greater than in working the pans. When ore is treated alone in the Woodworth Mill, the loss is $1\frac{3}{4}$ lb.

In retorting the amalgam, the retorts were charged with 1700 lb. of amalgam and the temperature gradually raised. Towards the last the furnace was made very hot and the fire then allowed to die out. The retorts were opened at the end of twenty-four hours. In the retorted amalgam a very singular separation was found to have taken place. On account of the large amount of copper sulphate used, the bullion is very base, especially in copper. The retort metal comes from the retort in the shape of a half disc, Fig. 184, which is divided into two distinct layers;



a - Copper.
b - Silver.
Fig. 184.

the upper one *a* is a more or less spongy mass of a reddish-brown colour, and comparatively brittle, which is very rich in copper, and contains the gold, if there has been any in the tails, and some iron. It is called the "base bullion" and averages about .070 fine, and rarely goes to .090. The lower part *b*, which has been next the body of the retort, is partially fused nearly white in colour, but with a slightly reddish tinge, contains the most silver, and is called the "white bullion." It averages about 450 fine, but is sometimes as low as 200. It was too dense to be crushed, and as it was generally nearly half silver and half copper, it was refined with great difficulty. Its impurities are copper and iron. As it comes from the retort, the two layers can be easily separated with a chisel. The base bullion amounts

to from two-thirds to four-fifths of the total weight of the retort metal. The proportion varied both with the amount of silver contained and the method of firing. A great many experiments were made to diminish the proportion of white metal in order to lessen the amount of sulphur to be used in treating it, but it was always found that doing so diminished the proportion of silver and the concentration of gold in the Doré bullion. It was always found best to retort thoroughly and produce as much fine silver as possible. From each cake taken from the retort three slices were taken, one from the middle and one from each end, with all their dirt, for assay. The samples, once taken, the base and the white metal were separated, and were sampled down to 6 lb. of white and 5 lb. of base metal, these being the charges for the small black-lead crucibles used. Each assay was repeated four times.

Annexed are a number of assays of this metal.*

EXPERIMENTS IN REFINING BULLION, 1873-4.
The "White Bullion."

Date.	Ounces.	Fineness by Assay.			Ounces contained.		
		Silver.	Gold.	Fine Metal.	Silver.	Gold.	Fine Metal.
Nov. 8, 1873 ...	2872.3	430.55	0.65	431.2	2755.8	4.2	2760.0
,, 15, ,, ...	3528.4						
,, 24, ,, ...	3557.5	433.0	0.5	433.5	2803.0	3.3	2806.3
Dec. 2, ,, ...	2916.0						
,, 9, ,, ...	2828.5	477.75	0.55	478.3	2633.0	3.0	2636.0
,, 22, ,, ...	2682.7						
,, 27, ,, ...	2624.4	485.75	0.55	486.3	1274.8	1.4	1276.2
Total	21009.8	...	...	...	9466.6	11.9	9478.5

After separation, the "white bullion" and "base" are generally treated differently. The white bullion was heated with sulphur in an old sulphuric acid parting kettle with a cast-iron cover, which was luted on to it. The kettle was charged with whatever

* These assays were made by Mr. Hodges, superintendent of the refining works of the Lyon Mill, and communicated to me at my request during the years 1874-1876. He has since published a detailed account of the whole process in the Transactions of the American Institute of Mining Engineers, vol. xiv., p. 731.

metal was on hand ready to be treated, at times as much as 450 lb. being charged, to which 18 per cent. of sulphur was added, and was heated to a low heat, the object being to form a sulphide of silver and copper. To facilitate its removal after treatment, old barrel staves were mixed through the charge. A small amount of sulphur escaped during the process, but generally as much as 16.7 per cent. combined with the bullion. The resulting sulphides were crushed and roasted to form copper oxide and silver sulphate. The amount of lumps in the crushing amounted to 6.86 per cent. The roasted material was leached with hot dilute chamber acid. The base bullion is broken and is roasted in a reverberatory furnace, with an iron hearth 8 ft. long and 5 ft. wide. The fireplace is 2 ft. wide, 4 ft. long, and 18 in. below the level of the hearth. The fuel used is wood. The charge is 400 lb. to 500 lb, which is the result of a day's work in the mill. It is charged at night in a hot furnace which has been used during the day for roasting stamp copper for the "blue stone" works, and left there until morning. It is then only partially oxidized, but is sufficiently so to be brittle and crush. The process was carried no further, as the metal is brittle enough to crush, and the oxidation has been carried as far as it can be economically done on large masses. The roasted bullion was first crushed in a small 5-stamp mill, and made to pass a No. 4 screen, and is then roasted in the same furnace for nine hours in the daytime, when the oxidation is made complete. A Chilian mill was subsequently used for crushing it. The cost of crushing was \$0.175 per pound for the white and \$0.088 per pound for the base bullion. As there was one part white to four parts base, the average was \$0.105 per pound of retorted bullion. The percentage of lumps remaining after crushing was not more than 0.55 per cent.

On next page a number of assays of "base" bullion after roasting are given.* These assays are the averages of from four to eight assays in each case. The "base bullion" was assayed after roasting, in order to get a fairer sample than is possible before roasting. There is no appreciable loss in this roasting, the dust chambers showing but little dust.

* Made by Mr. Hodges.

THE "BASE BULLION" AFTER ROASTING.

Date.	Ounces.	Fineness by Assay.			Ounces contained.		
		Silver.	Gold.	Fine Metal.	Silver.	Gold.	Fine Metal.
Dec. 9, 1873 ...	9389.5	67.15	1.95	69.1	630.5	18.3	648.8
,, 26, ,, ...	16038.0	70.05	2.15	72.2	1123.4	34.5	1157.9
Jan. 1, 1874 ...	11066.2	69.55	1.75	71.3	769.7	19.4	789.1
,, 6, ,, ...	15032.0	69.7	1.8	71.5	1047.7	27.0	1074.7
,, 9, ,, ...	10242.5	68.55	1.75	70.3	702.1	17.9	720.0
Total ...	61768.2	...	...	...	4273.4	117.1	4390.5
Total number of ounces in the bullion treated ...					13740.0	129.0	13869.0
Total number of ounces in the fine bars shipped ...					13678.7	127.9	13806.6
Loss in refining { ounces ...					61.3	1.1	62.4
Loss in refining { per cent. ...					0.45	0.85	0.46

From the roasting furnace the bullion of several days' roasting, or from 800 lb. to 1000 lb.—the products of the roasting of the two bullions being generally kept entirely separate—is charged into a lead-lined tank 5½ ft. in diameter and 3 ft. 9 in. deep, and a solution of chamber acid at 20 deg. Beaumé from the company's sulphuric acid works poured over it, steam being introduced into the liquid to make it hot. The acid, if it was pure, is not strong enough to dissolve the silver, but as it is chamber acid and contains some nitrogen, some silver, and all the copper, is dissolved. The liquid is drawn off and the silver is either precipitated from it with salt, or by boiling with copper from twelve to twenty-four hours. If the silver in solution is precipitated by salt, it is decomposed by zinc, and the silver after washing melted with the rest of the silver. If by copper, the copper sulphate is syphoned off and sent to the "blue stone" works. If strong enough it was at once crystallised. If too weak the copper was precipitated with iron. The cement copper was melted into bars and used to precipitate the silver.

In the tank there remains the undissolved silver and all the gold. This is transferred to leaching vats, 4 ft. long and wide and 2 ft. in depth, and the precipitated silver added to it, when it is washed until it is entirely "sweet." As the amount of

impurity is large it will require considerable washing. It is

COST OF REFINING 89,394 LB. RETORTED BULLION.*

——				Per Cent.	Per Pound Bullion.	Per Pnd. Blue-stone.
		$	$		cents.	cents.
Roasting, crushing, and sulphurising	Wages, 525 shifts @ $3.10 . . .	1629.96				
	Fuel, 86 cords @ $7.85	676.14				
	Sulphur . . .	60.94	2,367.04	15.1	2.65	0.79
Dissolving	Wages, 586 shifts @ $3.70 . . .	2166.50				
	Fuel, 240 cords @ $7.85	1883.63				
	Acid, 302,053 lb. @ 1.33 cents. .	4038.15	8,088.28	51.7	9.05	2.68
Concentrating	Wages, 27½ shifts, @ $3.50 . . .	96.25				
	Fuel, 77 cords @ $7.85	604.62	700.87	4.5	0.78	0.23
Crystallising and packing	Wages, 160 shifts @ $3.58 . . .	...	572.09	3.7	0.64	0.20
Cooperage	Wages, 76 shifts @ $3.61 . . .	275.03				
	Material . . .	115.20	390.23	2.5	0.43	0.14
Repairs	Wages, 76½ shifts @ $4.17 . . .	319.65				
	Material . . .	243.22	562.87	3.6	0.63	0.20
General	... ...	...	2,951.19	18.9	3.30	1.03
Total manufacturing cost			15,632.57	100.0	17.48	5.27
Copper 48,039 lb., paid for @ 15 cents . .			7,205.85	...	8.06	2.38
Total amounts charged			22,838.42	...	25.54	7.65
Wages, 1451 shifts @ $3.49			5,059.48	32.4	5.66	1.70
Material			7,621.90	48.7	8.52	2.54
General			2,951.19	18.9	3.30	1.03
			15,632.57	100.0	17.48	7.65

Of the total cost, including copper ($22,838.42), wages constituted 22 per cent. ; material, 65 per cent. ; general, 13 per cent.

washed twice a day for three or four days, for which reason the bullion must be left to accumulate for four days at least.

* Trans. Am. Inst. Min. Eng., vol. xiv., p. 755.

As the silver is in a very fine state of division it is dried by pressure in canvas bags, and melted in black-lead crucibles to refined Doré bullion. The refined bullion from the bags is .950 to .975 fine in silver, and .025 in gold. When the refined bullion and white bullion are melted together, as it sometimes was, the product is .550 to .560 fine in silver and .002 to .005 in gold.

The statement on page 463 of cost was taken from the Lyon Company's books for 1876 at a time when prices were unusually high. It gives, however, the details from which the cost of the treatment at the present time may be estimated.

The following Table gives the value and quantities worked in the tail mills of White Pine County, Nevada, in 1879.*

Name.	No. of Tons Treated.	Value per Ton in Dols.	Gross Yield in Dols.	Cost of Extraction. in Dols.	Net Yield in Dols.
Union Mill Co. ...	7,155	3.89	27,846.81	19,545.49	8,301.32
Lyon Mill & Mining Co.	35,329	3.58	128,177.00	116,555.55	11,621.45
Woodworth Mill Co....	11,518	5.53	47,154.12	48,515.38	...
French Mill	1,560	8.09	12,625.12	2,720.00	9,905.12
Atlanta Mill Co. ...	4,580	3.21	18,929.24	16,905.00	2,024.24
Eureka Mill	...	3.23	8,399.73	6,336.85	2,062.88
Pacific Mill Co. ...	12,150	12.47	149,644.15	83,856.00	65,788.15

One of the most difficult problems in the treatment of tails which has been successfully dealt with has been at the Tombstone Mill in Arizona.† The process which was adopted was concentration supplemented by fusion in a shaft furnace. The ore of the mine was at first composed almost entirely of silver chloride with a little sulphide. It ran 60 oz. in silver, ½ oz. of gold, and 3 per cent. of lead, with some manganese and iron oxide. With the depth of the mine the proportion of the silver sulphide increased and that of chloride diminished, and in addition some compounds of tellurium appeared in the ore, whose exact composition could not be ascertained, as it occurred as very minute specks scattered through the ore, which rendered the treatment very difficult. While the chloride remained in excess, neither salt nor blue stone was necessary, and 85 per cent. of the silver and 45 per cent. of the gold was saved, the bullion being 880 fine. When the ore changed, the use of these substances became indispensable. The bullion,

* "Production of Gold and Silver in the U.S.," Burchard, 1880, page 107.
† Reports of the Tombstone Mill and Mining Co. from 1881-1886.

however, would sometimes run down to 400 or 500 notwithstanding their use, the base metal being always lead. The mines were at Tombstone and the mill at Charlestown, ten miles distant. The ore was stamped wet, settled, and the pulp was treated in pans and discharged from the settlers into tail reservoirs, made by throwing up dams of earth. Nothing was to be gained by leaving the tails exposed to the air, and the ore was already thoroughly oxidized. This was shown by the fact that a second amalgamation of the tails yielded only $1.50 to the ton. Concentration on a large scale seemed to be the only practical method, as the experiments previously made had not been successful unless the ore contained at least 8 to 10 per cent. of lead, and those of this mine contained on an average but three, and even when their concentration had been made, there seemed to be no practical way of treating them, as there was no iron ore for flux to be had. The ore beside is extremely friable, and all the material which must be treated has been stamped through a 30 to 40-mesh screen, and has been reduced, to commence with, to a maximum of from one-fortieth to one-sixtieth of an inch in diameter to prepare it for the pans, where it is again ground for hours, which not only increased the proportion of fine materials up to 60 per cent., but made them sticky by trituration. Experiment showed that the tails could be concentrated either on the Frue Vanner or on German rotating round tables, but it was found in treating them that the very fine slimes ran over the tables on which they were to be concentrated without depositing the material containing the silver. This defect was subsequently cured by using buddles instead of tables. The tails of this mill do not generally carry more than 3 per cent. of lead, 50 per cent. of their weight being the finest possible slimes. It was supposed that the material resulting from the concentration treatment, was so very fine as to effectually preclude any success in working it in a furnace, but a careful trial proved that this idea was unfounded. There were found upon the property considerable amounts of manganese ore carrying 20 oz. of silver to the ton, which would neither amalgamate or concentrate, but which was used successfully as a flux for the fine slimes in the furnace, and not only regained the silver in the slimes, but that in the man-

ganesc which could not have been otherwise saved. While the percentage of silver extracted by amalgamation in the pans was reasonable, the amount of gold was very small. To raise this, as well as to gain, if possible, a larger part of the value of the tails, Mr. John A. Church made the following experiments, which have been placed as far as possible in chronological order, in order to show how the present method of working was arrived at. In 1881, after a series of preliminary experiments made for the purpose, six Frue Vanners were erected, and commenced to work regularly by the end of February, 1882. In January, 1883, two revolving buddles were built as tail machines to the Frue Vanners, and worked with such success that others were erected to supplement them. The method of working adopted was as follows: The richest part of the tail reservoirs were selected for the experiment; the dry tails were thrown into an agitator, where they were stirred with water, and the heaviest sands settling, giving the first concentrated product. It was richer than any other product in gold and silver, and contained enough lead to be smelted. What was carried off by the water was run on to the Frue Vanners, where a finer concentrate was obtained. After concentrating, the tails were raised in a belt elevator to the round table, which removed about one-third of the silver remaining in them. From this table the finest material, which still ran off without concentrating, was delivered to the second table of lighter slope, and from there to the third. The Vanners worked

		Assays, per Ton.			
	Tons.	Silver. oz.	Gold. oz.	Lead. per ct.	Per Cent.
Mill tailings treated	11,467	13.21	0.22	8.00	
Production:					
From the agitator	93	52.65	0.48	23.20	
From the Frue Vanners ...	1483	45.22	9.58	30.90	
Percentage saved by weight ...	...	...	...	...	13.31

Weight of tailings required to make 1 ton of concentrates, 7.5 tons.
Percentage saved, by value: silver, 41.29; gold, 34.09; lead, 50.61

Cost of Working.

				Tailings.	Concentrates.
Labour	Total	\$9,672.40	Per ton	\$0.843	\$6.34
Supplies	,,	873.13	,,	0.076	0.57
Total		\$10,545.53		\$0.919	\$6.81

perfectly so long as they were not pressed, but when the amount passed over them, reached as high as 5 tons per day; the very fine slimes were carried over them and lost. The necessity of having a large output, and the difficulty of handling them with such facilities as could be had, led to the reconstruction of the mill.

The results of the thirteen months' concentration are given on page 466. The value of the tails, in lead, was not determined by daily assays as the silver was, but the proportion of lead was estimated at 8 per cent.

The extraction of silver in the mill was about 80 per cent. and of gold about 45 per cent., and the treatment of the tailings increases this by 41.29 and 34.09 per cent., respectively, so that the final results are as follows:

Silver, extracted by amalgamation	80	per cent.
,, ,, concentration, .20 × 41.29 ...	8.26	,,
Total	88.26	,,
Gold, extracted by amalgamation	45.	,,
,, ,, concentration, .55 × 34.09 ...	18.7	,,
Total	63.7	,,

These experiments were not altogether satisfactory, but they eventually led to others which were eminently so, and as a result of these a concentration mill was built at Tombstone. The use of the round tables had proved so successful in catching the fine stuff, which, when the work was pressing, passed over the Vanners, and had shown so decidedly that the finest slimes could be treated on them, that it was decided to use them altogether in the new mill for such material, and to use jigs to treat the coarse stuff, and to size the material as far as possible. This mill was designed to treat the tailings from a 20-stamp mill, treating 40 tons to 60 tons a day as they ran directly from the pans in current work, and to mix with them at the same time the accumulation of old beds of tails. A launder, 1000 ft. in length, was erected to bring down the discharge from the settlers to the concentrating mill. This discharge runs in flushes from 40 to 70 times a day, each flush lasting about five minutes, and carrying from 2½ tons to 3 tons of ore and 1200 gallons of water. Between the flushes only a small quantity, and that of the finest material,

is allowed to run. To overcome the irregularity, an equalising box, 16 ft. long and 2 ft. square, was placed in the launder which holds back the excess and discharges it more slowly in the intermediate periods of light supply. The heavy material settled in the box. This was washed out with a stream of clear water, discharged with some force against it. In this way all the water for mixing the dry tailings passed through the equaliser. The dry tails were shovelled up by hand into the launder from the equaliser, and, in order to thoroughly mix the two, an agitator was used. The work of shovelling was stopped when the tails were sent down from the mill. The equaliser is a rapid running pug mill, formed by an octagonal box, 30 in. across, with fixed arms in the sides and a central shaft carrying a strong vertical basket of ¾-in. iron bars, which rotated within the iron rods projecting from the side of the box. The shaft revolves 105 times in a minute. At first very great trouble was experienced from the constant breaking of the iron rods in the basket, and the consequent stops and repairs, which at first seriously increased the cost of concentration. After a number of trials the side bars were shortened and the basket enlarged, when over 10,000 tons ran through it without a break. The agitator has worked up as much as 170 tons of the tails in a single day. It discharges into a 12-in. bucket elevator, which raises it 26 ft. to one corner of the top of the mill.

The mill itself consists of two sets of machinery, one for sizing and the other for washing the pulp. The sizing apparatus was in the upper story of the mill, and consists of two barrel screens and six separating hoppers. Four of the latter have a rising current of water, the remainder being merely settling tanks with discharge at the bottom. The last two of these had a capacity of 600 gallons each. They would fill up entirely during a flush, but the water would sink in the interval. During the flush there was a great increase in the discharge, which showed itself immediately on the machines. This was regulated as much as possible by stopping the shovelling of the dry tails, and by storing in the supply tanks, but there was always more discharge at this time than others. These sorting hoppers did not work well, because they did not separate the finest slimes from the

coarse sizes, and all the concentrating machines which were served by the hoppers received a portion of the slime, which followed along with the coarse material to the highly inclined tables, which were not suited for them. Every investigation made to ascertain the source of loss resulted in tracing it to the fine slimes that had never rested on the tables. Once deposited, there was no difficulty in washing them quite clean. The concentrating apparatus consists of two jigs, with two trommels with punched screens of $\frac{3}{64}$ in., seven German round tables, and ten dead ties. The jigs are of the ordinary Hartz pattern, which discharge through the beddings. They have a quick down stroke and a slow return, and made 120 strokes a minute, a style which is not now in vogue, but works excellently well on the Tombstone material. What passed through the trommels went to the line of hoppers. All that remained behind went to the first jig. The second jig was supplied from the first hopper. The second, third, and fourth hopper supplied the three round tables, and the fifth and sixth the others. The round buddles are 15 ft. in diameter with a slope from 7 in. to 4½ in. in 7½ ft., varying with the coarseness of the material to be treated. Four of these had iron frames and three were built of wood. Wood, in Arizona, is found to be sufficiently stable when the frames are heavy enough, and are covered with sheet iron before being cemented. All the surfaces are made of Akron cement. This surface is excellently well adapted to hold the very fine slime of lead carbonate which finds its way to all the machines in spite of the care taken to prevent it. The buddles revolve 105 times in 100 minutes. Brushes were not used, the material was cleaned, and the concentrates worked off with streams of water. Although covered with streams and jets of water, these buddles will retain slime that is too fine and light to settle in the tanks after leaving the tables. The Table on next page gives the details and cost of treatment at a time when the richest parts of the beds had already been treated.

With slower work much better results would have been obtained. Two jigs and six tables were expected to treat 120 tons a day, but they often did treat 150 tons to 170 tons, when the tails were so fine that the jigs did not do the work pro-

perly. Some of the tables treated a ton an hour regularly. The real saving was somewhat larger than the Table shows. The tails from the tables, on which the finest slimes were treated, were run through a series of six settling tanks, and gave a product containing 8 to 10 per cent. of lead, and 12 oz. to 15 oz. of silver. This was dried and used as a binding material for the bricks, but no account of it was made in estimating the work of the mill. The cost varied as the tail beds nearest the

April 1st, 1883, to March 31st, 1884.

		Old Mill.	New Mill	Total.
Days run		126	144	270
Tons treated		3,346	13,623	16,969
New tailings		...	6,150	6,150
Old tailings		3,346	7,317	10,663
Ore crushed		...	156	156
Product, tons		395.20	1,495	1,890.20
Ratio tailings to product		...	...	1:89
Per cent. saved:				
By concentrates	Gold	...	...	48.81
	Silver	...	...	40.57
	Lead	...	...	72.06
By tailings	Gold	...	...	55.53
	Silver	...	...	53.11
	Lead	...	...	77.61
Cost of treatment:				
Labour...		$4,375.50	$15,671.70	$20,047.20
Supplies		292.36	5,273.71	5,566.07
Total		4,667.86	20,945.41	25,613.27
Per ton, tailings		1.39	1.537	1.509
,, concentrates		11.81	14.010	13.550

mills became worked out and the distance of transportation increased. The first mill was run by water power and received a considerable part of the tails direct from the settlers, and was worked at a cost as low as 92 cents per ton of tailings. This increased subsequently to $1.39, and afterwards to $1.50. When round tables were used in the place of the Frue Vanners, the cost fell to $1.23, though at that time steam power was used exclusively, and the cost of its use was 24 cents a ton. As these figures relate to the experimental stages of the mill, there is no doubt that the cost can be still further reduced. It was found on carefully examining them, that the coarse grains were rounded

and tended to roll off the tables, so that Frue Vanners were afterwards exclusively used to treat this part of the tails with great success. It seems probable that the losses might be still further reduced, but very doubtful whether they could have been brought down lower than 25 per cent., except by crushing the coarse parts, which as well as the slimes contained the silver. The experiment proved, however, conclusively that fine slimes can be treated successfully on concentrating machines. The quality of the slimes was excellent; they contained about 50 per cent. of lead, but the percentage of silver varied with the richness of the material from which they were concentrated. Subsequently a seventh round table was used for the purpose of reducing the silica in the fine slimes used as a binding material in making bricks for the furnace with success.

As it was found that it was quite possible to treat the concentrated material in a furnace, in September, 1882, a furnace was erected, which was run for a short time only, as it was found impossible to get a sufficient coke supply. Several short runs were made with this furnace at different intervals, but with such success that the experiments were continued. There is nothing of special interest about the working of the furnace. It was 11 ft. high, with iron dust-chambers, which connected with a chimney 40 in. in diameter and 80 ft. high. It is the ordinary water-jacket furnace of the West. It started in May, 1884, and ran successfully until the cracking of the jacket caused it to be blown out. Until November of that year the breaking of these jackets gave a great deal of trouble, by either cracking near the top or burning out, and the stoppages were frequent and expensive. In November the cast-iron jackets were replaced by wrought-iron, and since then there has been no difficulty. The wrought-iron jackets were made of the same shape as the old ones, but on a different plan from the ordinary wrought jacket. The front and back plates were made of soft boiler-iron, and the edges of 4½ in. channel-iron. The front plate was put on with rivets, and the back with patch bolts. The water inlets were made of cast-iron, and were of the usual shape. These jackets have worked most satisfactorily ever since. The height of the stack has little by little increased to 30 ft. It was found that

large quantities of dust were lost in the escaping gases, so that it was necessary to make extensive additions to the condensing chambers. The tendency which the lead and silver had to pass into the slag when fine dust was charged without bricking was overcome entirely by changing its composition. The fuel used was Colorado coke; occasionally English coke was purchased. This coke was not always of good quality. The charcoal made in the country was so poor that it could not be used in current work, and was only used in blowing-in. To meet the difficulty presented by the sandy condition of the concentrates they were at first made into bricks by hand, using the fine slimes as a binding material. This work proved itself so successful, that a brick machine was introduced, in which the concentrates and flue dust were mixed with very fine slimes from the tail beds, moulded, placed on hacking pallets and dried in the sun, the only difficulty in their use being the lack of clay to be used as a binding material. The slimes contain 85 per cent. of silica, and only 2 to 3 per cent. of clay. They do not have the compressibility or binding power of clay, and owe what binding power they have to their extreme fineness. For this reason, burning does not improve the quality of the bricks enough to pay for the cost of doing it, and it was supposed that the possibility of running the furnace would depend upon the number of days of sunshine in a week. This uncertainty, however, has not proved the impediment that it was supposed it would, for repeated trials have shown that this very light material, so fine that the breath will blow it out of the hand, can be smelted even without bricking. When charged in improper proportion, the sand sometimes pours out of the tuyeres unaltered, but when the furnace is properly managed they can be easily worked. The great objection to the use of sand as such is not the difficulty of treating it, but the large quantity of flue dust made, so that the sands alone are not treated, if the weather has been clear enough to have a stock of sun-dried bricks on hand. When the furnace was blown in, no lead could be had to protect the hearth. It was therefore blown in on concentrates that had been specially prepared for the purpose.

As the lead in the ore was carbonate, there was no difficulty in its reduction. The main trouble was the fine condition of the material to be treated. The low fusing point of the manganese which had

to be used as a flux, it being the only substance suitable for the purpose found, had the disadvantage that it allowed the less fusible material to settle out of it and rapidly fill up the hearth. The almost complete absence of sulphur in the tails treated caused speisse to form, which generally ran out with the matte, but, if the flow was stopped, this cooled almost immediately, and formed engorgements which it was almost impossible to melt. The use of manganese as a flux after a number of trials proved, however, successful, and, as it contained some silver, added to the value of the product. When the charge was very basic 50 tons to 55 tons could be treated in a day and without danger of engorgement. With a more acid charge there was less difficulty, but only 40 tons could be treated. The composition of the slags varied from day to day, but they were always clean, and very free from both lead and silver owing to the manganese they contained. The usual composition of the slag is given below:*

Silica	29.60
Manganese	43.25
Protoxide of iron	11.56
Lime	7.50
Alumina	6.34
Lead	1.40
Total	99.65

The furnace work from September 1, 1882, to March 31, 1883, consumed the following materials:

	Tons.		Percentage of Charge.	
Concentrates	438		23.65	
Tailings	438		23.65	
Ore	47.92		2.70	
Manganese	625.25		33.80	
Silver bearing material ...		1549.17		13.80
Limestone	33.02		1.80	
Slag recharged	260.80		14.40	
Cleanings recharged	6.00			
Fluxes		299.82		16.20
		1848.99		100.00
Colorado coke	168.17			
English coke	81.00			
Charcoal...	67.84	317.01		17.15
Total materials ...	2166.00 tons.			

* "School of Mines Quarterly," vol. v., p. 219.

Production, 2708 bars; 144.88 tons, containing—

	Silver. oz.	Gold. oz.	Lead. lb.
In bullion shipped, 131.95 tons	40,883.57	298.81	263,333
,, on hand, 12.93 tons	4,654.80	34.00	25,537
	45,538.37	332.81	288,870
Market value of bullion shipped	$44,170.18	$6,013.22	$9,068.52
Total value	...	$59,251.92	
Freight and charges	...	4,189.61	
Net return	...	$55,062.31	

The value of this product is as follows, the return from the bullion on hand being estimated:

Returns from 121.95 tons shipped	$55,062.31
12.93 on hand	6,000.00
Returns from 93 tons concentrates amalgamated	5,500.00
Total product from re-working tailings	$66,562.31

The lead bullion made was sent to refining establishments in the East. It was of good grade, fair softness, and carried a much higher proportion of gold to silver than the ordinary mill bullion. As a considerable part of the silver was derived from manganese ores without gold, the results of the processes have shown that this method is particularly valuable for the saving of gold.

The concentrates sent to the mill during the year 1882-3 contained more quartz and less lead than the average product. They milled up to about 65 per cent. of their value, and were subsequently reconcentrated. In preparing the concentrates for the furnace they were mixed with about half their weight of fine slimes from the oldest bed of tails. It was expected that the whole of this bed, which is by far the richest portion of the tails, could be smelted without any concentration at all, but it subsequently proved that it was too poor in lead, which prevented its being done. The difficulty experienced in operating the furnace when the attempt was made to utilise the concentrated tailings was so great that it seemed to be insurmountable, but it was overcome as shown above. The amalgamation of the concentrates proved so unsuccessful that it had to be abandoned, and the success of the operation was found to be dependent entirely upon furnace work alone. It was found that by merely stirring up the tails with water and floating off the fine part, a sandy residue was obtained of .08 of one per cent. of the whole mass of tailings,

but yielding in the mill $5500, as has been previously shown. This amounts to 48 cents per ton of tailings. Changes that are necessary in the treatment, such as using a small proportion of concentrated tailings, prevented any close reliance upon the results obtained at first, but the figures below give the results obtained; 3800 tons of tailings were concentrated during the year at a cost of $7.69 a ton, including fluxes and ores purchased for smelting with them. The yield after smelting was found to be $16.05, or a profit of $8.36 per ton of tails. The cost includes the cost of construction, mining, and all other possible expenses. The results obtained in the first treatment of the tails were not uniform. The sandy portion concentrated best at first, but experience in the treatment led them eventually to do almost as well with the other parts of the ore. The result of twelve months' workings was the obtaining of 1480 tons of concentrates from 10,417 tons of tails. Of these concentrates 438 tons, or 29$\frac{6}{10}$ths, were smelted with an equal weight of tails, so that the quantity of tails represented in the furnace product was 3540 tons. This yielded $61,000 or $19.61 per ton at a cost of $9.42 per ton. The amount of lead in the charge is so small that they were not able to treat a mixture so silicious as that which would have resulted by using equal weights of concentrates and tails.

The Table on page 476 gives the operations of the furnace for the years 1883 and 1884. The quantity of fuel reported includes waste in handling, blowing in and out, and all other items.

In the table of production the quantities given are distributed by shipments and not by actual current product. The difference is small, and in a yearly statement only affects the closing month.

The cost for the same period, the furnace having treated 7764 tons, is given below:

	Total.	Per Ton.
Labour	$29,724.80	$3.83
Supplies	44.714.93	5.75
Ore hauling...	10,636.89	1.37
Bullion hauling	1,290.94	17
Repairs and renewals	1,214.13	15
Flux purchased	18,651.78	6.25
Ore purchased	41,436.73	
Lucky Cuss mining	10,689.26	1.38
Total	$158,359.46	$18.90

The product of the furnace from April 1st, 1883, to March 31st, 1884, was:

	Weight.	Contents.			Value.			
		Silver.	Gold.	Lead.	Silver.	Gold.	Lead.	Total.
	Tons.	oz.	oz.	lb.				
May ...	54.425	13,548.76	134.527	107.910	$14,926.63	$2,780.68	$4,597.13	$22,304.44
June ...	79.437	23,987.70	204.309	157.225	26,431.44	4,223.06	6,380.49	37,034.99
July ...	53.836	19,547.03	136.080	106.324	21,519.37	2,812.77	4,199.80	28,531.94
August ...	13.518	5,976.98	29.330	26.625	6,567.21	606.25	1,051.69	8,225.15
September	68.027	17,044.06	61.220	134.882	18,853.79	1,265.40	5,287.39	25,406.58
Six months	269.243	80,104.53	565.466	532.966	$88.298.44	$11,688.16	$21,516.50	$121,503.10
October...	55.067	15,066.53	52.034	109.097	16.625.17	1,075.53	4,022.57	21,723.27
November	68.722	13,702.93	41.487	136.501	15,104.71	857.52	4,633.95	20,596.18
December	54.867	20,008.74	95.657	108.357	22,139.28	1,966.89	4,028.85	28,135.02
January ...	81.842	30,690.33	170.021	161.569	34,353.36	3,514.32	5,885.88	43,753.76
February	13.651	3,242.11	15.426	27.079	3,639.26	318.85	1,029.00	4,987.11
March ...	109.521	30,745.63	239.014	276.100	33,759.98	4,920.22	8,113.06	46,793.26
Six months	383.670	113,456.17	613.139	758.703	$125,621.96	$12,341.49	$27,713.31	$165,988.60
Year ...	652.913	193,560.76	1178.605	1291.669	$213,920.40	$24,341 49	$49.229.81	$287,491.70

TOTAL PRODUCT OF FURNACE.

	Silver.	Gold.	Lead.
	oz.	oz.	tons.
November, 1882—March, 1883	40,883.59	297.86	131.67
April—September, 1883	80,104.53	565.47	266.48
October, 1883—March, 1884	113,456.17	613.14	379.35
Total...	234,444.20	1476.47	777.50

FURNACE.

April 1st, 1883, to March 31st, 1884.

		Materials used.			
		Tons per Charge.	Total.	Percentages of Charges.	Total.
Days run	279				
Number of charges ...	21,829				
Weight of charge		0.390	8,512.152		
,, fluxes		0.083	1,815.580		
Concentrates slums and flue dust		0.137	2,999.000	0.352	
Ore		0.052	1,130.082	0.132	
Manganese		0.161	3,511.000	0.412	
Limestone		0.006	124.070	0.014	
Slag and cleanings...		0.034	748.000	0.092	100.2
American coke		...	1,269.180		
English ,, ...		...	464.000	...	21.329
Charcoal		...	52.400		
Product:					
Number of bars		...	11,851		
Shipments, tons		...	654.470		
Containing silver, ounces... ...		...	193,560.7		
,, gold, ,, ...		...	1,178.6		
,, lead, tons		...	645.84		

When silver ores or tails, as they frequently are called, are composed of minerals, some of which are, and others are not, attacked by mercury, and the ore is so poor as not to bear the expense of roasting, they can very often be concentrated to advantage both before and after, or either before or after the treatment in the pans. It was formerly considered that concentration of the pan tails was always more advantageous than the concentration of the ore, but under some circumstances, as in the case of the Montana Company, both are practicable, though it will generally be found most advantageous to concentrate the pan tails, when the ore contains silver chlorides, but it will sometimes be wise to concentrate the ore, especially when the amount of sulphurets is large. It is apparent that this concentration of low-grade ores loses all of its advantages if there is any great increase in the cost of the treatment, and that when it is done it must be done by a treatment as nearly automatic as possible, and in conjunction with such processes as the Boss continuous amalgamation system, and at a very low cost. The concentrates must be very rich and in small amount. Sometimes they are not more than one to three per cent. of the original ore. After concentration

the whole of the tails, if they are worth it, can be passed to the regular pan amalgamation. They will not be so rich as the ore before amalgamation, but will frequently be rich enough to treat. There are also some rich ores which have, under certain local conditions, been concentrated before amalgamation, and have yielded better commercial results than those gained by the more expensive process of roasting. How to treat these concentrates, when obtained, depends largely upon circumstances. The treatment of a very small amount of rich concentrated material is a very different matter from that of working the whole original quantity of ore. It will often be found best not to treat these concentrates at the works where they are produced, but to ship them elsewhere. In certain cases it may be worth while to erect cheap furnaces for fusing them with other ores, or to roast and amalgamate them. When they can be leached raw it will often be wise to use some of the leaching processes for a part of the treatment, and to concentrate the tails if they are worth it. The difficulty in concentrating tails has always been to get a continuous supply for the concentrators, hence the experiments made at Tombstone to overcome it. With the ordinary system of settling in tanks which is used in most of the silver mills, it is almost impossible to arrange for the regular feed for the concentrators on account of the discharge of the settlers being made at irregular intervals. This, however, might be done by arranging the operations of the mill so as to discharge the settlers at regular times, and in this way to get a constant flow of the tails without using too great an amount of water, but it necessitates a considerable amount of labour and trouble in charging and discharging the pans. It can only be successfully carried out in large mills, and in a small one would be too complicated and expensive. In such mills a remedy could be found by the introduction of the Boss continuous system. The most recent experience has shown that this process will give at least as good a result as the old tank system which was so many years in use, and that it affords a great economy in handling the pulp. On all such ores as that of the Montana Company, the advantage of removing the sulphurets from the ore before it is treated in the pans, is clear. The tails from the concentration can either be

CONCENTRATION MILL OF THE MONTANA CO., TREATING 105 TONS DAILY OF THE TAILINGS FROM 50-STAMP WET CRUSHING SILVER MILL.

Fig. 185.

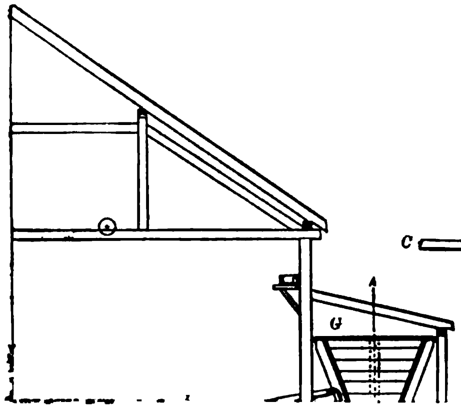
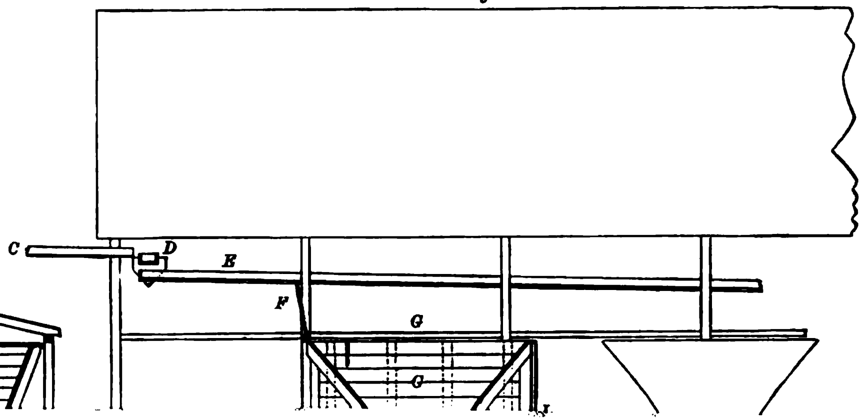


Fig. 186.



run into the old tanks to be settled and treated in charges in the pans, if the mill is an old one, or, better, settled in spitzkasten, as is now done in some of the modern mills, and treated continuously by the Boss system. Some of the more modern mills and some of those now being built are arranged with elevators, so that either concentration and amalgamation, or the reverse, may be used, according as experiment shows one or the other to be most advantageous.

The Montana Company's mill at Marysville, Montana,* is an excellent example of the operation of these principles. This company has three mills, one of 60 stamps treating gold ores, and one of 50 and one of 10 treating silver ores. The tails from these latter mills were first treated by amalgamation and then by concentration, as it was found possible here as well as at Tombstone to profitably concentrate them before the second treatment. The tails from the pans were brought to the concentration mill by a launder C, Figs. 185 to 187, in the centre of one side of the concentration mill, and from here carried to distributing launders D, which are connected with others, E, at right angles to them and running the whole length of the building on either side. These launders E, as shown by the arrows, are divided so that each spitzkasten receives one-fifth of its supply, which is brought down by the pipe F into the very large pointed boxes G, where the settling is effected and the discharge on to the Vanners K, effected by means of the launder J from the syphon in the spitzkasten. The overflow water from the boxes on the opposite end of the feed is quite clear. It runs through a pipe I into the underground sewer H. The pulp discharge J is divided into two, so that each spitzkasten supplies one Vanner. When the spitzkastens are properly constructed the pulp can be discharged on to the Vanners K, thick enough for good working, with a loss of fall of not over 2 ft. The percentage in the tails of the Vanners is so small that it is not worth consideration. About November 15th, 1866, this company decided to change their process for silver ores, so as to extract more of the precious metals by concentration both before and after amalgamation.

* For this information and the cuts I am indebted to W. F. McDermot, of New York.

The ore was first crushed wet and then amalgamated on copper plates, then concentrated, and the concentrates either sold to the smelters or roasted and amalgamated in pans, the tailings from which are concentrated and again treated. Continued experiments have shown that the ores of this mine can be treated much more profitably by the pans after concentration than before. Twenty Frue Vanners were placed in the 50-stamp mill, and four in the 10-stamp mill at a cost of building and machinery of $16,500. After nine months' work the twenty-four Vanners yielded $349,139, which was all profit except a very small cost of treatment. All experiments on these ores have shown a result of from $8000 to $10,000 in favour of concentration before amalgamation. The following is the output of the mine in both gold and silver for the year 1886:

	Tons Crushed.	Yield in Concentrates.	Yield in Bullion.	Total Yield.
Ten-stamp mill	5,007	$68,734.25	$247,505.43	$316,239.68
Fifty-stamp mill	33,171	280,405.82	1,032,942.45	1,313,348.27
New sixty-stamp mill (36 days run)	3,550	9,764.56	32,068.09	41,832.65
Vanner house (tailings from fifty stamps) ...	......	23,711.75	......	23,711.75
	41,728	$382,616.38	$1,312,515.97	$1,695,132.35

It has been found recently desirable, as the ores contain a large quantity of sulphurets, to change the order of treatment, concentrating first and then treating in the pan. The advantage of doing so is shown in this mill by the fact that the pulp, being free from sulphurets, the amount of mercury lost in the pans, which was formerly 1.55 of a pound to the ton of ore, was immediately reduced first to .8 of a pound and then to .62. The bullion at the same time was brought up from 550 to 930 and even 950 fine; there has also been a much smaller quantity of slag from the crude bullion fusion than formerly.

Concentration after amalgamation has been practised for a very long time, but its application to low-grade tails like those at Tombstone and elsewhere is quite new. It is not profitable except on the condition of having machinery which is as nearly automatic as possible. When the Boss system has been perfected there are many large bodies of low-grade ores as well as tails that can be worked by it and concentration at a cost of from $2.50 to

$3 per ton. In the condition in which they come from the mines many of these ores cannot be commercially amalgamated, both because of the loss of precious metal and of mercury. Many of these ores can be concentrated, or where they cannot be their tails can, so that either by concentration and amalgamation, or *vice versâ*, such ores can be made to pay. In some cases it is even possible by either one or the other of these treatments to work ores which could not be crushed dry and roasted, as the extra cost of doing so would more than equal the gain in doing it. Such treatment has been carried out extensively at the Silver King Mine in Arizona since 1877, although there is a roasting plant on the ground. At these mines the ores run from 40 oz. to 60 oz. to the ton. Since 1877, 4-ft. Vanners were used for the purpose of concentration. In the year 1882, the very unusual amount of 92 per cent. of the ore assay was recovered by this method. The value of these concentrates was $1094 per ton. In the year 1883, the amount recovered was 89 per cent. of the assay. In August, 1886, six 6-ft. Vanners were started in the mill. By January, 1887, a period of four and a half months, they had treated 10,178 tons of tails from the twelve 4-ft. Vanners on which the first concentrations were made. The average amounts treated on each of the 6-ft. Vanners was 12½ tons per day, which yielded 4.11 oz. of silver per ton. These were concentrated up to 45.48 oz. The quantity of material treated was 480.46 tons, and effected a saving of 21,509.47 oz. The tails from the large Vanners yielded only 2.03 oz., or 7½ per cent. of the value of the original ore, and was almost entirely composed of argentiferous zinc blende. The second class concentrates contained 24.68 per cent. of blende, 2.23 of galena, and 33.35 of barium sulphate. The surplus water of the original pulp, like that of the Montana Mill, was carried off in pointed boxes. This method of concentration, either before or after amalgamation, seems likely to have a very great future in the West.

CHAPTER X.

LEACHING PROCESSES.

For a number of years it has been evident that, with certain classes of ores, neither the smelting nor the amalgamation processes have given satisfactory results. That either the loss of the precious metals was too great, or that the process itself was too costly to allow of the treatment of rebellious or poor ores. For this reason attention has been turned to leaching; and especially to the Von Patera process, which has been successfully introduced in a number of works for treating ores to which no other process seemed at the time to be applicable. But even this process fails where the quantity of base metal, especially lead, increases beyond a certain amount, as these metals are dissolved by the hyposulphites, and are consequently precipitated with the precious metals, causing a loss of the reagent and making the bullion too base. None of the older leaching processes answer the purpose where the quantity of lead is large. When ores or tails can be concentrated so as to give a product rich enough to smelt, the problem seems very simple. The tails, if they are rich enough, can then be submitted to some lixiviation process; or when, as is the case at the Old Telegraph Mines, silver is contained as chloride in oxidized ores having sufficient lead, they may be first crushed, then leached for silver, and the lead and other minerals, if they are worth it, concentrated from the leached product. How to treat an ore which cannot be concentrated, and which contains too little lead to smelt and too much to lixiviate, has been one of the most difficult of the metallurgical problems.

In the simplest case the lixiviation processes in use require that the silver in the ore should be transformed into a chloride, since metallic silver, or any other of its compounds, except the chloride, are but slightly soluble in sodium or calcium hypo-

sulphites. Hence there is always a loss in silver, as it is impossible so to conduct the chloruration that all of the silver shall be converted into chloride. How to extract more of the precious metals, or how to separate that contained in the tails, has been the important question which seems to have been at least partially solved by the Russell* process, which proposes a solvent for the compounds of silver other than the chloride, which acts only slightly on this salt, so that after leaching out the chloride the tails may be treated. In some cases the ore is not chlorurised, but only roasted to drive off the excess of sulphur, or in the case of slightly rebellious or of oxidized ores they are not roasted at all, but treated directly. It has been found that the lead dissolved from the roasted or unroasted ore can be separated from the solution as lead carbonate without the loss of any of the other metals. This discovery is most important, and its application is very easy.

What is and always has been needed is a process applicable to impure or poor ores, or to both poor and impure ores, as these are those which are most commonly found. The prices of labour, fuel, and transportation, together with the richness of the ore, will always determine whether smelting processes can be employed, but these we suppose to be out of the question. The problem is how to treat an ore to which neither fusion nor amalgamation, either on account of the insufficiency of the capital or the actual or probable richness of the tails, render it inexpedient to treat by any of the usual processes.

Von Patera Process.

When the ores do not contain much base metal they can be treated by the Von Patera process. This process of leaching, which is based on the solubility of silver chloride in sodium or calcium hyposulphites, did not attract much attention in the West for some years, partly because imperfect experiments made with it in a small way, had not been successful. It had also been thought that, while the price of salt is very low in these regions, it would be impossible to use any amount of a reagent which was high-priced like sodium hyposulphite. As far

* Trans. Am. Inst. Min. Eng., vol. xiii., p. 47.

as the chemicals are concerned, while the price of salt is very low, all the salt, in any process where it is used, is lost. This expense is, therefore, a considerable one when very large quantities of ore are treated. Though the price of sodium hyposulphite is high, the amount consumed is extremely small, since all but a very small portion of the liquid is saved, as most of it is regenerated and used over again, the cost of this reagent will be very low. There are very few places in the West where lime cannot be had. The use of calcium sulphide, which is so easily made as a precipitating reagent, makes it quite possible to use this leaching process, as this substance gradually transforms the sodium hyposulphite into calcium hyposulphite the use of which has a great advantage in the treatment of ores containing even a very small quantity of gold, as the calcium hyposulphite dissolves nearly the whole of the gold, and allows of its being extracted, while the sodium hyposulphite is not so efficient. The quantity of water used with pan amalgamation must always be at a maximum, even though it is used over again. The quantity used with hyposulphite leaching is always a minimum, since all the water used in the process can be used over and over again, even the washing water being serviceable, so that the loss of water will be very small. In addition to this the plant which is to be used is a very cheap one, being composed of roasting furnaces, which need not be of a very expensive type, of wooden tubs, of not very costly materials, and requiring for the most part only low-priced labour. The process, however, requires careful watching by an expert, and continual assays, in order to see that there is no waste of silver nor of the reagent.

Besides this, the California practice invariably associates with the pan the California stamp, which has always been considered one of the best machines for crushing. In the case of surface ores, however, especially such as contain silver, either in a native state or as chlorides or bromides, or where they contain silver sulphide, the stamp is a very bad machine, because it tends to beat out the pieces so thin that they float, or in case of brittle ores to make flour, and in this way permits of their being carried off by the water. Later European practice shows that this has such an effect in enriching the tails that rolls are there gradually

taking the place of stamps. The rolls simply crush or disintegrate the material, and are much less expensive than the stamps. But even supposing the stamps to be replaced by rolls, the rest of the amalgamation plant—the furnaces, pans, and settlers—is costly, requires constant repair, and must, in a period more or less short, wear out and be replaced. The mercury is, too, an expensive and troublesome reagent. The consequence is that the capital required for a leaching plant is very much less than it would be in a milling one. The leaching process is also applicable to ores containing both gold and silver, for when sodium hyposulphite is used after the ores have been leached for silver, the tails can be treated by Plattner's process, and the gold and silver both recovered in a state of high bullion, so that a parting process would not be necessary, and when calcium hyposulphite is used they are recovered together. It is also applicable to ores very rich in silver as well as to very poor ores, whether they are or are not very impure or are contaminated with other metals, since, when it is worth while to do so, small amounts of copper, cobalt, and nickel may be separated.* There is, however, a limit to the quantity of base metals, especially lead, which can be treated. This will depend in every case on the quantity of silver and on the cost of reagents. It is never applicable to ores which contain lead enough to smelt. In some cases where the ores were very rich in pyrites but poor in gold and silver, a matte concentration has been made and the extraction done on the roasted matte. Such an application necessitates a cheap fuel, but the concentration can be carried on so as to materially reduce the amount to be treated. In Mexico† this process has been used on amalgamation tails, containing large quantities of lead and only 11.5 oz. or 0.24 per cent. of silver. Where no lead is present, much smaller values have been treated in the United States. If the amount of sulphur in the ore is very small, it may be necessary, in order to get high chloruration, to mix it with pyrites. It has been found that highly oxidized ores containing manganese will give high chlorurations when they do not contain any sulphur. Whether it is desirable to treat the ore raw or

* "Annales des Mines," Series 5, vol. viii., p. 68.

† "Zeitschrift für das Berg-Hütten und Salinen-Wesen," vol. xxi. (1873), p. 143.

after chlorurising or oxidizing roasting must be determined in every case by experiment; but whenever a furnace is used, the dust in the dust chambers must be well chlorurised, as the silver in it would be otherwise lost, but the fine material will become all the greater impediment in the leaching.

It is not, however, to be supposed that the process has no disadvantages. While the plant is very inexpensive, it requires careful attention on the part of those in control of it, for although the reactions are exceedingly delicate they can be learned by men of very ordinary capacity, provided they are properly superintended; but the least carelessness on their part, either in the roasting, leaching, or precipitation, or by adding too much or too little of the reagent, involves very serious losses.

The process has assumed great importance of late years from its introduction in so many works, and especially from its use at the Old Telegraph and Lexington Mills, the works at Triumfo, in Lower California, and from the erection of a large plant recently at the Geddes and Bertrand Mine, in Secret Cañon, near Eureka, Nevada, where a poor ore full of impurities is treated. In the description of the process as used in these localities no mill in particular has been described, though most of the details refer to the Bertrand Mill.

The analysis of the ore from the Bertrand Mine is given below:

Silicic acid	50.25
Iron	8.06
Zinc	7.62
Lead	4.64
Arsenic	0.73
Antimony	1.35
Silver*	0.17
Lime	4.92
Magnesia	2.40
Sulphur...	0.96
Carbonic acid	8.30
Water	3.80
Loss and oxygen	6.80
Alumina	trace
Bismuth	,,
Copper	,,
Potassium	,,
Sodium	,,
	100.00

* About $50 per ton. Most of the ore is, however, of a lower grade than this.

The process consists of seven different operations:

1. Crushing the ore.
2. Drying the ore.
3. Roasting it with salt.
4. Leaching out the base metals with water.
5. Leaching with hyposulphite of soda.
6. Precipitating the silver.
7. Roasting the sulphide of silver and melting for bullion.

I. Crushing the Ore.

The ore of the Bertrand Mine comes from a higher level than the mill. It is brought in wagons drawn by horses and is dumped into a tunnel leading to the mill, falling through a shoot into cars on a track running into the highest level of the mill. Eventually a tunnel will be run directly to the mine, which is about a fourth of a mile distant, and the ore will come to the mill without previously discharging.

From the cars the material is dumped upon a grizzly, which is an inclined iron grating allowing only the small pieces to pass and sending the large ones directly into a crusher, which after breaking them up discharges them into the same bin into which the small pieces which passed through the grizzly have fallen. From this bin the ore falls through a shoot into cars which carry it to the dryers. In some works the large pieces pass through two sets of crushers, and what passes through the grizzly goes into a second crusher set fine, into which all the ore which does not pass the screens also falls. The ore is crushed so as to pass a 15 to 20-mesh screen; 30-mesh screens were first used, but it was found that the material did not discharge from these as well as from a coarser mesh, and that there was no necessity of treating the ore finer, as the roasting and leaching were better done on the coarse ore. Experience has shown that with the coarse screens more ore can be treated in a given time, as it leaches faster and there is less fine material to clog the filter. With fine ore it sometimes takes six or seven days to leach, and even then it is imperfectly done. In making an examination of the effect of coarse and fine screens it was found that in using those with from twenty to forty meshes, 31 per cent. of the ore passed through;

from 40 to 60-mesh screens, 14 per cent.; from 60 to 80-mesh screens, 6 per cent.; and finer than this scarcely an appreciable quantity passed, without mechanical agitation such as comes from the blow of the stamp or the agitation of the screen. The size adapted to each ore can only be determined by experience, as ores which are apparently the same act differently in leaching. The limit of coarseness depends on the character of the ore, and the way the silver-bearing minerals are distributed in the gangue. The only general rule that can be given is that the ore must be crushed just as coarse as is consistent with perfect chloruration* in the furnace, which can easily be determined by trial. This question has received but little attention. It has more importance than is generally attributed to it, and when improperly done easily translates itself into both a diminished output and a loss of money. It may be said in general, that a No. 10 wire screen is the limit of coarseness which it is advisable to use, that in most cases a screen between 16 and 20 will be the one which can be used to the greatest advantage, but that No. 30 will rarely be needed.

The ore of the Bertrand Mine, wet before passing through the dryers, assayed, September 28th, 1882, \$26.71; on September 29th the assay was \$29.85, and on the 30th, \$23.85. These assays were taken from a large car into which a sample from every mine car is thrown. The mean of these three is \$26.80, which is a little low, the net assays being about \$30. They are given because the other assays are made on charges made the same day.

II. Drying the Ore.

The ore is damp when it comes from the mine and is taken from the crusher to the dryers. These are revolving wrought-iron cylinders, 20 ft. long by 4 ft. in diameter at one end and 3 ft. at the other, known under the name of Pacific dryers. The iron-work for the dryers weighs about 10 tons; they are not lined. The flame from the fireplace runs directly through them, the ore being fed at one end automatically by the Hendley's Challenge

* The word chloruration is used to describe the formation of chlorides by means of salt, in contradistinction to chlorination, used to describe the formation of chlorides by means of chlorine gas.

Automatic Feeder, and dumped out into cars at the other end of the dryer, without manipulation. These dryers are usually heated by a fireplace of their own, which, however, is not absolutely necessary, as the flames from the Brückner cylinders might be made to pass through them and then be made to enter the dust chambers, thus utilising a large amount of waste heat. When only small samples of ore are to be treated they are carried to a special bin, and put through a Dodge crusher, which is used almost exclusively for sampling. Drying floors, made by passing the waste heat through flues covered with cast-iron plates, are used in some works. This saves the fuel used in the dryers; but this economy is more than compensated for by the labour required, the dryers being automatic in their action.

After the ore leaves the drier it is carried to a bin, from which it passes over a 15-mesh screen, and falls through a shoot in which is arranged a system of magnets to catch any pieces of metal which may have accidentally got into the ore, either in the mine or in the mill, such as bits of broken picks or drills, as they would be likely to injure the rolls if they were allowed to pass through them. The ore then passes through two sets of Krom rolls, 16 in. by 24 in., from which, all that passes through the screens is carried by a chain elevator to a storage bin in the upper part of the building. What fails to pass the screens is carried back and put through the rolls again. The ore from the rolls assayed, on September 28th, 1882, $27.75; on September 29th, $25.13, and on the 30th, $24.19.

III. Roasting the Ore with Salt.

From the storage bin the ore descends through a shoot into cars standing on a track scale, where it is weighed. The contents of the cars are dumped into a hopper above the Brückner cylinders, the amount of each charge passing into the hopper being carefully weighed. The moment the hopper is discharged into the cylinder beneath another charge is put in. The salt is not weighed. It is measured in soap boxes which contain about 80 lb. each, and is mixed with the ore either in the dryers or in the hoppers; formerly 5 per cent. of the weight was mixed with the ore.

This amount was gradually decreased until now only 3 per cent. is used.

A number of experiments have been made as to the best place to add the salt. It was for a long time the practice to add the salt in the battery, but it was soon found that heavy stamps were not well adapted to either crushing or mixing the salt. It was then added in the hoppers, and became thoroughly mixed by the movement in the cylinders. Now it is added in the dryers, and by incorporation resulting from the movement there and in the rolls it has been found that the quantity of salt may be considerably reduced, so that they now do not use more than a third of the salt they formerly did. Very extensive experiments have been made in Europe on the best place to add the salt, in the various metallurgical works where salt is used for the extraction of the metal, which has resulted in the adoption of a very ingenious mixing machine, into which the ore and salt are charged, which has produced great economy in the use of salt and better subsequent working.* Both methods are successful, but the introduction in the dryer seems the best, as it takes the place of the mixer in the European methods. At first 2 per cent. of iron pyrites was mixed with the ore in order to insure a proper roasting. The quantity was diminished little by little until now none is used. In most cases, however, where there is a large amount of base metals this addition will be necessary.

All the conveying of the ore is done with chain elevators having pockets 6 in. by 4 in. These are used for the dry ore only, the chloridised ore is not elevated. Repairs to these chain elevators are very easy, for when a link is broken it has only to be taken out and another one put in, or if for any reason it is desirable to make the chain shorter or longer the links can be readily removed or added.

The ore is now ready to be roasted. This may be done in any kind of a furnace. Where transportation is difficult a reverberatory furnace, with a hearth arranged in three steps, so that the ore in passing from one to the other falls a distance of 4 ft. to 5 ft., would be the best. A Stetefeldt furnace could also be used to advantage. The ironwork of this furnace, as

* Trans. Am. Inst. Min. Eng., vol. xiv., p. 101.

it is in pieces of no very great weight, can be easily transported, but is more expensive to build than a reverberatory furnace, which can always be easily adapted to any kind of fuel. At the Bertrand Mill, which is within easy reach of San Francisco by railroad, there are four Brückner cylinders, which are 7 ft. in diameter and 19 ft. long, and hold a charge of about 5 tons. The fireplace is on a prolongation of the axis of the furnace, but was formerly put at right angles to it, greatly to the inconvenience of the workmen. The cylinder is driven by friction rollers, of which there are three sets, and not by a gear-wheel round the body of the cylinder as in the older form. The work is continuous, the furnace never being allowed to become cool. As soon as a charge has been treated a fresh charge is immediately put in. To introduce the charge the manhole is brought under the hopper and its valve drawn. It is then replaced and the cylinder set to revolving two to three turns per minute. The amount of sulphur contained in the Bertrand ores is exceedingly small, so that the salt in very small quantity, if it has not already been added in the dryers, may be introduced at once. When ores containing a large amount of sulphur are used, a careful roasting at a low temperature must precede the chloruration, steam at a low pressure being introduced for the purpose not only of getting rid of the last trace of sulphur, arsenic, and antimony, but also to decompose the chlorides of the volatile base metals, the nascent chlorine thus given off acting very energetically on the silver. When the ore contains base metals which it is desirable to save,* this roasting must be done with great care, and the value of the base metals separated must compensate for the extra expense in fuel owing to the use of steam in the roasting. If lead is present, the roasting must be done at a low temperature, for as the compounds of lead are easily fusible, there might be danger of agglomeration; or, if the temperature is high and silica is present also, a lead silicate might be formed which would prevent the solution of the silver. Special care must be taken in such a case to transform all the lead into chloride, as this is soluble in hot water, while the sulphate is not.

The roasting lasts eight to eleven hours, depending on how the

* "Annales des Mines," Series 5, vol. viii., p. 70.

charge works. When it is finished the man-hole is opened without stopping the cylinder, which in its rotation discharges the ore, which falls, at the Bertrand Mill, into pits cut out of the rock in the foundation, just underneath the cylinders, where it is allowed to remain about nine hours, until it is ready to go to the cooling floor. These pits have been found to be a very great advantage, for it has been ascertained that a considerable amount of chloruration takes place in the pit after the charge leaves the furnace, so that the time of waiting is not lost. It is red hot when it falls into the pits but cools sufficiently to be drawn off into cars after some time. Occasionally the ore, when for any reason there is a stoppage, remains for two or three days in this bin and is still hot when drawn. Generally, however, it is drawn out on the cooling floor as soon as possible, where it is at once moistened with water to keep down the dust. Sometimes the ore is put into the tub so hot that the water boils, but this is not usual. The ore is generally cold enough not to make any appreciable difference in the temperature of the water. When the ore is one which is not habitually treated, a sample is drawn through the fireplace in order to test the chloruration. When, however, the ore is that which they are constantly using, they recognise that it is finished by its rolling about in the furnace with a sluggish motion somewhat like that of damp sugar.

When the furnace is discharging two assay samples are taken from every charge. When about half the charge of the furnace has run out a long iron spoon is run underneath and filled and its contents assayed. One of these samples is used for a chloruration test, which is made at once for each charge, each assay sample being marked with the number of the charge and of the furnace or the cylinder; the other is thrown into a large iron car to be afterwards used in making the general assay to ascertain the amount of silver contained in the ore.

Occasionally the charge of ore from the furnace is more than the tub will hold. This residue is put into a heap, and when enough of it has accumulated to fill a tub it is leached and called a "mixed charge."

On September 28th, 1882, the ore from the cylinder assayed $32.99; on September 29th, $30.47; on September 30th, $27.96.

These assays on the same charges and days are interesting, and are given below together.

	Sept. 28.	Sept. 29.	Sept. 30.
Damp mine sample	26.71	29.85	23.88
Sample from the rolls	27.65	25.13	24.19
„ „ Brückner's cylinder ...	32.99	30.47	27.96
„ „ „ „ ...	32.36	...	30.32

Eleven assays of the ores taken from each of the furnaces during different days are given below:

ASSAYS.*

No. of Charge.	No. of Furnace.	Value.
74	1	$25.44
75	1	30.16
81	3	32.36
81†	3	31.42
84	2	28.27
85	3	33.30
86	3	27.96
115	4	30.16
116	4	27.96
117	4	25.75
118	4	27.33

The assay for chloruration is made on the sample taken from the charge; 15 oz. are weighed out and put into a funnel with a proper filter. Sodium hyposulphite from a tank above is let on to it and allowed to run from fifteen to twenty minutes, the filtrate being collected and sent to the leaching tubs. When no silver is dissolved the assay is washed and dried, and a fusion assay made of the tails. Twenty-three such assays are given below.

CHLORURATION ASSAYS.

No. of Charge.	No. of Furnace.	Value.	No. of Charge.	No. of Furnace.	Value.
81	3	$5.49	111	2	$3.92
81‡	3	5.20	112	2	3.61
101	1	5.18	113	2	3.77
102	1	4.40	114	2	4.24
103	1	2.98	184	4	3.29
104	1	4.40	185	4	4.08
105	1	3.77	186	4	3.14
106	1	4.71	187	4	3.45
107	1	3.14	188	4	3.45
108	2	3.77	190	4	4.55
109	2	3.45	198	4	3.29
110	2	3.61			

* I am indebted for these and the following assays, to my former pupil, Mr. C. F. Pearis, who was for some months assayer at the Bertrand Mill.

† After remaining two days in the bin. ‡ After remaining two days in the bin.

The following Tables give the details of twenty-four charges in the cylinders during a period of six days. They give a complete record of the work of the furnace during that time. The corresponding Table for the tubs, and also the chloruration and tub-tailing assays, are given on pages 507 and 509. These Tables comprise the details of the entire work of the mill for four days.

Number of Furnace.	Number of Charge.	Ore. tons.	Ore. lb.	Salt, lb.	Pyrites, per Cent.	Cars of Flue-Dust.	Hours Roasting.
	Mixed charge	5		450	2		
2	67	5	532	720	2	3	10
	Mixed charge	5		450	2		
2	68	7		720	2		7¼
2	69	7	803	720	2		8
	Mixed charge	5		450	2		
2	70	7	1041	720	2	4	11
2	71	5		720	2	3	9½
2	72	6	1758	480	2	5	9¼
2	73	7	50	720	2		8¾
3	74	7	140	700	2		13
3	75	5	600	480	2		7¾
4	88	5	1642	640	2	2	7½
4	94	7	440	720	2		10
4	95	6	1116	720	2	1	8¾
4	96	5	1835	720	2	2	8½
4	97	7	140	720	2		10
4	98	4	134	720	2	3	9½
4	99	7		720	2		10½
4	100	3	667	640	2		9¼
4	101	7	1800	720	2		9¾
4	102	7	220	720	2		8½
	Mixed charge	5		450	2		
	" "	5		450	2		

The Brückner cylinders are run by an upright engine with a cylinder 10 in. by 12 in.

When the ore in the collecting bins is sufficiently cool, it is drawn into cars and taken to a brick cooling floor, where it is dampened with water and spread out with hoes. The collecting bins are 12 ft. deep and extend from the bottom of the cylinders to the top of the car on the floor below. The works are built on the side of a hill, which is a very convenient arrangement. No attempt is made at the Bertrand Mill to keep ore ahead. It is the intention to have two charges on the cooling floor and one charge in each of the furnaces and a charge in each of the hoppers. The roasting is done by two men on each shift, each

man tending two cylinders, with two Chinamen bringing the wood which is used as fuel.

All the machinery other than the Brückner cylinders is run by an engine with a 14-in. cylinder with a 48-in. stroke, which is much larger than is required for the work. Twelve cords of wood are burned in the four Brückner cylinders in twenty-four hours. The wood is cedar, nut-pine, and mountain mahogany, which are all excellent woods. They are equal to or even better than the lignites which are the usual fuels of the country, judging them by weight. The dryers burn three cords of wood per day, the engine four, the stoves to heat the leaching room one. Twenty cords of wood are used for the twenty-four hours for the entire work of the mill. One engineer on each shift runs the boilers and the engine. He is a Chinaman and has one helper, who is also a Chinaman.

A considerable amount of silver chloride and of dust containing silver is volatilised or mechanically carried off from both the Brückner cylinders and the dryers. It is, therefore, necessary to have dust-chambers connecting with both of these. The flues have a down-take, which communicates with a large chamber by an arched opening, which is quite small. The dust accumulates in considerable quantities at the bottom of the down-take, so that they are obliged to clean it frequently. This material and that which comes from the dust-chambers is collected. It is then passed through the Brückner cylinders and mixed with the charges as shown in the Table, page 494. Fresh salt is added to it and it is put through the tubs. Sometimes, when large quantities are on hand, it is treated separately.

IV. Leaching the Base Metals with Water.

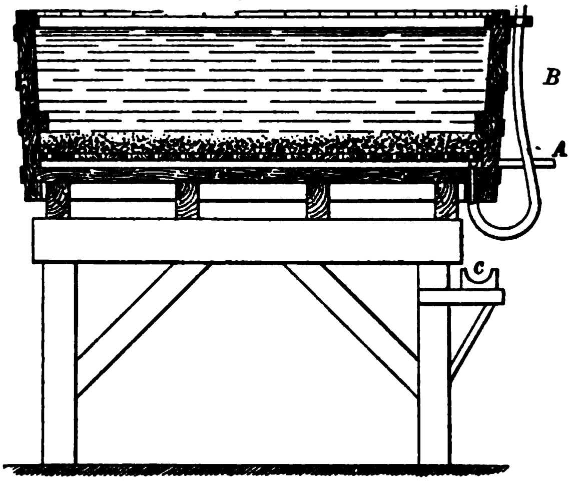
The leaching-room contains twenty-four tubs, Fig. 185, twelve on a side, each of which is 6 ft. in diameter and 3 ft. deep, the depth being regulated according to the facility with which the water will filter through the ore, and also to the ease with which a man can throw out the tails. It would be much better to increase the diameter of the tubs to 12 ft. or 15 ft., as is done in many of the works in Lower California and Mexico, and to increase the depth to 6 ft. 6 in. if the tails are to be removed by

hand, or 5 ft. if they are to be sluiced. Such a tub will hold 35 tons to 50 tons of raw ore, or from 25 tons to 40 tons of roasted ore. The staves should project beyond the bottom, forming a chine of 6 in. Tubs of any other size would answer as well so far as the leaching is concerned, but the ore can be treated with the greatest rapidity and with least expense in large-sized tubs. Experience in these countries has shown that a better result is obtained with large than with small tubs. The bottom pieces should be grooved and jointed with a tongue $\frac{5}{8}$ in. by $1\frac{1}{2}$ in. All the joints must be accurately made and the tub put together with a thick coat of white lead. The hoops must be made so that their ends are fastened together with screws which can be tightened when necessary. The discharge pipe is generally $1\frac{1}{2}$ in. in diameter. It is made as shown in Fig. 188, or is inserted into the middle of the bottom of the tub, and is screwed on to a cast-iron flange bolted to the bottom. It is made of lead or hard rubber. To it the rubber hose is attached which connects with a Körting injector. These tubs are set in many mills on the floor together in pairs, the two tubs almost touching each other, with a railway upon one side and an easy passage-way between each pair of tubs. This method of placing the tub is a bad one. They and all the other tanks should be placed on a platform, as is shown in Fig. 188, and this should be sufficiently high to be easily visited in case of a leak. The track runs close to the side of the tubs. That the discharging may be done with facility, when it is done by hand the wagon which receives the tails comes just above the top of the tubs, so that a man standing in them can easily throw the tails out into the wagon. When the tails are sluiced out they are easily disposed of.

Permanently fixed above the tubs at the Bertrand Mill are hoppers, which are long and narrow. They are divided into two compartments, and have at their bottom a slit valve, so that the ore may be discharged from each compartment in little piles over all parts of the bottom of the tub. Each tub may have its own hopper, made of iron, as at the Bertrand Mill, or one hopper may answer for two or more tubs, the discharge being made by flexible pipes. This arrangement, which is the first one adopted, is not a good one. It is better to have over the tubs a slightly inclined track

which comes from the cooling floors, if the ore is roasted, or from the ore bins if it is not. The ore is brought in this car, coming down by gravity, and is dumped into the tubs. Stop-cocks, unless made of hard rubber, or plugs, do not answer for the leaching tubs, as they are apt both to get out of order and to leak. A rubber pipe B, Fig. 188, is much easier to manage than a valve, and can always be securely fastened to the bottom or side of the tub, and when not in use can be hung up out of the way. The ore is passed from the cooling-floor to the vats as rapidly as it can be handled, each charge of ore being drawn from the furnace bins to the cooling-floor as soon as the latter is empty. The cars from the cooling-floor run over the top of the hoppers and dump their contents into them, which from

Fig. 188.



there fall into the tubs when the slit-valve is opened, and the ore is afterwards evenly distributed over the bottom of the tub with a hoe, or better the cars discharge directly into the tubs. The charge could as easily be put into the tubs from a car running on a railroad at a sufficient height above the tub not to inconvenience the workmen in discharging it, and arranged with a valve like the hopper, or it could be dumped into the vat from the side.

The bottom of the tubs is covered with four or five wooden slats, 3 in. by 1 in., which do not touch the sides, but leave a

space 1½ in. wide all round the tub. Over these, at right angles, are arranged other slats from 1 in. to 1½ in. apart. Canvas duck or gunny-sacking which is wet from the previous charge, or is wet purposely when a new filter is to be put in, is, in some places, placed over these. It is generally brought close up against the sides of the vat, so that no ore will pass, by means of a hoop, which fastens it securely there. Sometimes a strip is fastened to the sides of the tub in which a V is cut, and into this the ends of the cloth are packed. In some of the recent works the false bottom is 1½ in. from the sides. A hoop is put around it, leaving ½ in. between it and the sides of the vat. Into this opening the ends of the cloth are put and packed in with a hemp rope. In this case the discharge pipe must be in the centre of the bottom of the tub. Any or all of these methods are good. The ore falls from the hopper or car upon the canvas and rises to within 2 in. or 3 in. of the top of the tub. When the ore is once in the tub, special care is taken that it shall not be disturbed in any way. Once ready for the water it is allowed to remain without being disturbed until the tails are ready to be discharged. Any interference with the arrangement in the tub increases the difficulty of leaching. The simple pushing of a stick two or three times down through the ore may delay the leaching several hours.

In the bottom of the tub are two india-rubber pipes, about 1½ in. in diameter, one for introducing the water and the other for discharging it. When hot water is used for leaching out the base metals the ore is always wet from the bottom, the water being introduced through the pipe and coming up through the ore. It has been found that this method of moistening the charge causes the ore to cake less and the solutions to percolate through them much more easily than when the water was introduced from above. When the water or the solution is turned on, the level of the ore sinks. Raw ores and tails sink about 5 in., chlorurised ores sink from 9 in. to 15 in. As soon as the water completely covers the ore the supply is cut off. After remaining there for a very short time, the discharge pipe, which has up to this time been hung up higher than the top of the tub, is lowered and the water allowed to run out into the launder C; the leaching water is added at the same time. This water

is permitted to flow away entirely or is collected, according to the quantity of water available. When the leaching with water is finished, the water is discharged from the tank until it reaches the top of the ore. The discharge is then stopped and the hyposulphite solution turned on. The soluble matter removed by the first leaching will generally be sodium, chloride, and sulphate. If the ore contains any of the other metals, there may be besides contained in the leach water the aluminium, calcium, iron, manganese, zinc, and copper sulphates and chlorides. If there is an excess of salt, some of the lead and silver salts may be dissolved, and in such cases bottom leaching should be used and the silver precipitated. It will be mostly in the first wash water. As soon as sodium sulphide shows no reaction, the water is cut off.

The water used for the base metal leaching may be hot or cold. If the ore contains a large amount of lead which has been converted into chloride, it will be best to leach with cold water until the larger part of the lead and the surplus salt has been dissolved out. It is then leached with hot water, cooling the ore, however, with cold water before the hyposulphite is added, in order to prevent too great an extraction of the base metals with the silver. When the ore filters slowly it will be found best to heat the water, which will not only make it filter more easily, but will dissolve the base metals more rapidly. When the charge comes warm from the cooling floor, when introduced into the tub, the water grows warm from its heat. In such a case, unless the ores are very pure, or where very impure ores have been leached with hot water, the charge must be cooled with cold water before introducing the hyposulphite, or the bullion would be much more impure.

As the water introduced from the bottom subsides, a very thin crust is formed upon the top of the charge, which is carefully removed and put by itself until sufficient accumulates to be treated. This material is quite rich in silver. It contains all the silver which was dissolved by the excess of salt or other chlorides in the ore, and which would have been lost if the hot water had been introduced at the top. This amount is all the larger if the solutions are hot, or if the excess of salt is large, as a hot brine dissolves

more silver according as it is hotter and saturated, while a cold one dissolves hardly any. The dilution of the liquor with water precipitates part of the silver near the top and distributes the rest of it through the ore, so that but little is lost in bottom leaching. The top crust is collected in barrels ; there is but a small quantity on each tub, but, as there are twenty-four tubs constantly in use, it amounts to considerable by the end of the month. After this has been removed clear cold water is allowed to run in at the top, the quantity being regulated so that the inflowing water will just be equal to the quantity discharged, in order to leach out the base chlorides which are soluble. Any salt in excess, or any which has not been decomposed in the roasting, will be dissolved at the same time. When the ore contains but little base metal there is no advantage in using hot water. This is the case at Triumfo, in Lower California, where the ore is always leached cold and the water introduced from the top.

When the ore, however, contains a great excess of lead or antimony, the method of introducing the water from below is not sufficient, as the chlorides of these metals also are precipitated from the solution by dilution with water. In such a case the water is introduced from the top and the liquid allowed to flow from several tanks into compartments filled with shavings, so as to get a large amount of surface. Clear water is allowed to flow into these compartments, so as to make the liquor very dilute. The moment the fresh water touches the stream containing the chlorides, which was formerly clear, it becomes cloudy, and then precipitates the chlorides. Flowing over so large a surface, and being obliged to pass under one compartment and over the other, the silver, lead, and antimony chlorides deposit on the shavings, and can be dissolved from them with hyposulphite and the solution put with the other silver solutions. The water running off contains the zinc, copper, and iron. At Triumfo the ore contains 4 per cent. antimony and but little lead. No attention is paid to the antimony, the ores being leached with cold water. In Mexico the washing is repeated, the ore being discharged from the first tub to be put into a second. The first washing lasts for four hours and the second a little less; the observation having been made that even when no metal salts are found in the wash-water of the

first washing, some are found in that of the second; and only when nothing is precipitated from the second washing is the leaching with hyposulphite begun. In the West but one washing is generally made. When raw ores are treated, the rapidity with which the water will pass through the ore generally depends upon the quantity of slimes they contain, especially if they are clay or talc. This difficulty may be very serious where raw tails are treated. Roasted ores generally leach better than raw ores. When the filtration is too slow it may be hastened by the use of the acid syphon pump, which is used in most of the recently constructed works. It is a Körting injector, which is so made that it can be used as an ejector as well. It is used for all the solutions. By its use the water or the solutions may be exhausted from the bottom and returned to the top of the vat, so that the same solution filters through the ore continually. This is called circulation. The rate of leaching for raw ores, not forced, will be about 1 in. per hour. For chlorurised ores it will be from 1½ in. at the beginning to 1 in. at the end. It has been shown by experience that rapid filtration answers best. It should, however not be accelerated if it reaches to 6 in. or 8 in. per hour. It should be all the faster if the ores contain no caustic lime. In introducing the water or the solutions on the ore, the force of the current must be broken by discharging it against boards placed so as to scatter it, or by running the water or solution through a box, the bottom of which is filled with holes. The quantity of water required will be from 20 to 80 cubic feet per ton of ore; the time from 1½ to 4 hours.

When the water flowing from the tubs no longer gives a precipitate when tested with calcium sulphide, the base-metal chlorides are removed. When there is plenty of water, as at certain seasons of the year, all of this water is allowed to run to waste, if the ore contains no nickel or cobalt or other metals which are worth collecting. When, however, the water is scarce, or the base metals are valuable, it is all collected in the vats. The base-metal chlorides are precipitated with calcium sulphide and the water used over again. It takes a quantity of water equivalent to three tubfuls to leach the base-metal chlorides out. The addition of the water causes the ore to settle and diminish in volume 10 to

18 per cent. if the ore is raw, or 12 to 24 per cent. if it is roasted. The vat should, therefore, always be filled in such a way that the top of the washed ore should be at least 12 in. below the top of the tub.

V. Leaching with Sodium or Calcium Hyposulphite.

The washed ore is now leached with a cold solution of sodium hyposulphite, which is made in the storage tanks, which are provided with steam coils to heat the water and make the hyposulphite dissolve more rapidly. The strength of the solution will depend on the richness of the ore and the quantity of base metals present. If the ore is very rich and but little base metal is present, it may be used very strong, and even warm; but when base metals are present too much of them would be extracted, so that the solution is usually made weak and used cold. Experience, however, has shown that it is not desirable to wash with solutions richer than 2½ per cent., even for the richest ores. With low grade ores a 1 per cent. solution is sufficiently strong. This is made by adding 62½ lb. of hyposulphite to every 100 cubic feet of water. When starting it is usual to commence with a 2 per cent. solution, and then diminish until that suited to the ore is arrived at. At Triumfo the quantity used is generally one pound to eight gallons of water. At the Bertrand Mill it contains from half to three-fourths of a pound to the gallon. The hyposulphite is usually purchased. It comes to the works in small kegs, containing from 50 lb. to 60 lb. each, and does not usually cost more than 2.5 cents to 3 cents a pound. It is generally cheaper to purchase it than to make it. The liquors increase in quantity as the solutions are constantly being regenerated, so that but small additions have to be made, and this only to keep up the strength of the solution, as the liquors are constantly being diluted. The hyposulphite is likely to become impure when the ores are not properly leached with water to dissolve out the sodium sulphate and sodium chloride; it is generally the practice then to regenerate it by spreading wood ashes over the ore and leaching through that, the alkalies in the ashes taking up the sulphates and the excess of chlorides. By the constant use of the calcium polysulphide it gradually

becomes converted into calcium hyposulphite. The solution after a time may become caustic if the roasted ore contains caustic lime, or if sodium sulphide is used for the precipitation of the silver it may contain some caustic soda. The presence of only one-tenth of 1 per cent. of caustic soda reduces the solvent power sometimes as much as 30 per cent. At the Ontario Mill it was found that .5 of 1 per cent. of caustic lime reduced the solvent power from 11 to 24 per cent. As soon, therefore, as their presence is ascertained, they must be neutralised by the addition of sulphuric acid in the precipitation tubs after the precious metals have been drawn down. The liquor must then be tested with litmus paper. When 50 to 60 tons of ore per day are required to be treated, the amount of solution that must be constantly on hand will vary from 1500 to 2000 cubic feet. This is based on the calculation that it takes 9 cubic feet of solution to saturate one ton of ore. This quantity includes all that is in the vats in use and waiting to be pumped into the reservoirs, and that already there.

It is sometimes desirable, especially when the ore contains gold,* to use calcium hyposulphite, especially as the calcium polysulphide which is used for the precipitation of the silver is the first step in the process of manufacturing it. This is done by boiling 1.5 parts of the purest freshly slaked lime that can be had with one part of crushed brimstone. Flowers of sulphur sifted through a fine sieve is also used, but it does not answer so well as the brimstone, which can be more readily obtained and is more easily manipulated. If the lime is not very pure it may be desirable to use two parts of lime to one of sulphur. When sulphur is scarce and high-priced it is sometimes collected from roasting the silver sulphide. The water is first boiled with steam, the lime is added and well stirred, and the sulphur is then introduced. Sufficient water must be added to keep the mass liquid, but not enough to allow of the solution of polysulphide becoming too much diluted.

* Some laboratory experiments of Mr. Russell, Trans. Am. Inst. Min. Eng., vol. xiii., p. 90, seem to show that gold is no more soluble in calcium than in sodium hyposulphite. It has, however, been remarked in the works, that whenever calcium hyposulphite is used, the gold is extracted, and this does not always appear to be true when sodium hyposulphite alone is used.

The boiling is kept up from three to four hours. During this time calcium polysulphide mixed with calcium hyposulphide, is formed, as shown by the formula below:

$$3\,CaO + 12\,S = 2\,CaS_5 + CaS_2O_3.$$

This operation requires some care. If lime is added in excess insoluble compounds are apt to form. Sometimes calcium bisulphide is formed, which crystallises out as reddish-yellow crystals. An excess of sulphur, however, does no harm, and may be useful in the next operation. The liquid is allowed to settle and the clear liquid decanted. It is desirable that this solution should be from 8 deg. to 10 deg. Baumé, and only sufficient water is added to bring it up to that strength. The liquid is now treated with sulphurous acid, made by boiling charcoal broken to about the size of a grain of corn, with sulphuric acid of about 1.80 deg. Baumé, enough being used to make a pasty mass. This is done in an iron retort, adding the acid to the charcoal as it is necessary. There is no necessity of purifying the gas. When the yellow colour of the liquid disappears the sulphide has been converted into hyposulphite. This is tested with a dilute solution of silver chloride. So long as there is any precipitate or cloudiness the passage of the sulphurous acid must be continued; when there is none the solution is ready. There is an objection to this salt when hot liquors are to be used in leaching, which is that at about 60 deg. Cent. calcium hyposulphite decomposes into gypsum and sulphur, while sodium hyposulphite is not affected by boiling. When hot leaching is to be done, this latter salt is the only one which can be used.

The residue in the tub is now treated with the same amount of water as before, and the operation commenced over again. If the resulting liquor is not more than 3 deg. to 4 deg. Baumé, it is too weak, and is kept to treat the next charge of lime. The process used at Triumfo is much simpler. The residues left in the tub after the clear sulphide liquid is decanted are simply shovelled out and left for three or four days exposed to the sun. They are then leached with water, which extracts the calcium hyposulphite, after which they are thrown away. This method is a very rough one, as the oxidation is necessarily very

imperfect, but for that situation is a much more economical method than the use of sulphurous acid.

The calcium sulphide made in the first stage of the process is the material used for precipitation. It should not be less than 6 deg. Baumé if used for that purpose. In some works the sodium polysulphide, made by boiling soda with sulphur and treating it with sulphurous acid to make sodium hyposulphite, is used. This reagent does not precipitate the silver so rapidly, nor so well, and is not so easily washed. For this reason the calcium hyposulphite is preferred. Sometimes sulphuretted hydrogen, made by melting paraffine and flowers of sulphur together, is used for the precipitation. This is much more disagreeable than the other method and is not so frequently used. Where sodium hyposulphite is used, the constant addition of lime transforms the solution gradually into calcium hyposulphite. The strength of this solution is constantly kept up with a hydrometer.

When the ore is very impure, it is generally best to keep the solution at about ½ deg. Baumé. Before turning the solution into the precipitating vats, care must be taken to see that the water used for leaching out the base metals has been displaced. This is easily done by tasting, as the hyposulphite of soda and silver has an intensely sweet taste, or better by testing the liquor flowing from the tanks with calcium sulphide. As soon as the least turbidity is shown, it is time to catch the liquor, as the hyposulphite is acting. This hyposulphite liquor is allowed to run through as long as it has a sweetish taste. No special attention is paid to the time at the Bertrand Mill, as the work is done by Chinamen whose records would not be very intelligible or trustworthy; tests only are relied on. The reaction which takes place is

$$2\,AgCl + 2\,Na_2S_2O_3 + 5\,H_2O = 2\,NaCl + 2\,NaAgS_2O_3 + 5\,H_2O.$$

This sodium silver hyposulphite is exceedingly soluble. The quantity of hyposulphite, and also the time required for leaching, will depend on the richness of the ore, more time being required for a rich than for a poor one. Ores are rarely treated that require a longer time than twenty to thirty hours. An ore containing from $300 to $400 a ton] will be perfectly leached in

twelve to fifteen hours. The time required at the Bertrand Mill is from six to twenty hours, depending on the ease with which the hyposulphite filters.

As soon as the hyposulphite ceases to taste sweet, the solution is tested with calcium sulphide to ascertain whether the ore is exhausted of silver. The tester carries a small bottle of the sulphide solution in which he has a stick. He takes a tumblerful of the liquid running out and lets a drop or two of the sulphide fall into the liquor in the glass. If there is a precipitation of silver sulphide, the lixiviation is continued. If there is no precipitate, and the liquor becomes only slightly discoloured, he puts in some of the liquor containing the silver solution to ascertain whether there is an excess of calcium sulphide. In this case the hyposulphite solution is discontinued and the excess of hyposulphite must be washed out with cold water. This is done by turning on the water as soon as the hyposulphite reaches the surface of the ore. Water to saturate the charge is added, and when it in its turn reaches the surface of the ore, the discharge is turned into the fresh-water tanks, or is let to run to waste. The quantity used will generally be from 7 to 10 cubic feet per ton of ore, and the time about 1½ to 2½ hours. The exhaustion of the hyposulphite is distinguished by the taste. This last liquor, as it contains nothing, goes to the fresh-water tanks.

It takes from eighteen to forty-eight hours to charge, leach, and discharge the ore, the time depending on the way the roasting has been done. Exceptionally a charge will take a longer time, sometimes as much as five or six days. In Mexico the leach liquor is divided into two parts, as it has been found that that coming off first contains much less of the sub-sulphates and oxychlorides of the base metals than the last, and consequently produces a purer bullion. This is not done in the West, and, when all the product must be cupelled with lead is not necessary.

The Table on next page gives thirty-seven assays of the tub tailings made at the Bertrand Mill during four days. They show a very low value in silver.

As soon as the water ceases to flow, a probe sample is taken from three points in each tub, if the tails are to be sluiced out,

or a sample taken from each car if they are shovelled out; these samples are sent to the assay office to see whether there is any silver left. The ore assays from 30 oz. to 50 oz.; the tails should assay only about 4 oz. These assays are being con-

TUB TAIL ASSAYS FROM AUGUST 28TH TO SEPTEMBER 1ST, 1882.

No. of Charge.	No. of Furnace.	No. of Tub.	Value.
			$
76	2	7	3.14
81	3	4	6.59
87	2	17	4.87
89	2	2	4.71
90	2	5	4.71
92	2	24	4.55
94	4	14	4.08
98	4	2	3.14
99	4	4	4.08
101	4	8	4.08
103	2	2	3.92
104	2	6	4.08
105	4	1	3.61
106	4	14	3.92
107	2	11	4.08
108	4	6	3.77
109	2	16	3.29
110	2	6	4.71
132	3	19	3.61
135	3	3	4.40
138	3	15	5.18
140	3	1	1.08
141	3	7	4.24
143	3	14	3.92
144	3	21	4.08
146	3	9	4.24
147	3	17	4.24
148	3	3	3.92
150	3	21	4.08
172	4	4	4.40
173	4	8	4.87
174	4	12	3.45
175	4	22	3.29
176	4	20	4.55
178	4	10	4.55
180	4	5	4.55
183	4	20	3.29

stantly made, twenty to thirty being often made in a day. If the assays show that the silver is down to from 4 oz. to 6 oz., the tub is discharged, if not, the ore has to be re-roasted and treated with the hyposulphite again. This rarely happens,

however, for, as the assays are constantly being made, the exact condition of the ore is known before the leaching commences.

The ore from the tubs at the Bertrand Mine is shovelled into cars and run out to the dump-heap at the rear end of the mill, where there is a turn-table to switch the cars into the proper track. Care has to be taken at first to see that the men do not cut the gunny-sacking in shovelling out the ore. At first, before the men are experienced, it is generally considered a necessary precaution to cover the gunny-sacking with slats from 5 in. to 6 in. apart, to insure that it is not cut by the shovels. When, however, the workmen are experienced, they know when they are near the bottom from the height to which they have to throw the leached ore, and they are so careful not to dig on the sacking that these slats need not be used. The sacking is very seldom cut after the men are accustomed to the work. In the recently constructed works, the tails are discharged from an iron gate painted with asphalte and fastened to the outside of the vat. The bottom of this gate is ¼ in. below the top of the filter. The gate is of iron, protected with india-rubber. It requires about 12 cubic feet of water per ton to sluice out the ore. If water is scarce, the wash water after the silver has been removed may be used for the purpose. Sluicing is a much better, more convenient, and cheaper method of removing the tails.

The amount of water required for sluicing tailings usually averages about 12 cubic feet per ton. In case water is scarce the tailings must be removed by other means, and the expenses per ton will be increased about 20 cents. The total amount of water required in the metallurgical treatment of chlorurised ore and raw ore and tailings in each of the cases cited on pages 538 and 539 for which estimates have been given, is as follows:

Chlorurising and Lixiviation of very Base Ore.

1st.	If the tailings are sluiced	102 cub. ft.
2nd.	,, ,, not sluiced	90 ,,

Chlorurising and Lixiviation of Average Ore.

1st.	If the tailings are sluiced	62 ,,
2nd.	,, ,, not sluiced	50 ,,

Raw Lixiviation of Ores and Tailings.

1st.	If the tailings are sluiced	22 ,,
2nd.	,, ,, not sluiced	10 ,,

Raw Lixiviation combined with Concentration.

1st. If the tailings are sluiced ... ... 22 cub. ft.
2nd. ,, ,, not sluiced ... ... 10 ,,

In the concentration of the tailings an additional amount of water is required, which varies considerably according to the method employed.

The following Table gives the details of the leaching of the same charges, the details of which, during the roasting process, are given on page 494.

No. of Furnace.	No. of Charge.	Hours on Water.	Hours on Soda.	Total Hours Leaching.	Assay.	Chloru-ration.	Tub Tail-ings.	No. of Tub.
	Mixed charge	11½	49½	61	25.44	4.08	4.71	1
2	67	9⅚	19	28⅚	42.73	3.45	3.67	16
	Mixed charge	11⅔	25½	37⅙	28.89	8.79	4.24	17
2	68	11⅙	17	28⅙	46.18	6.12	3.92	19
2	69	10	15½	25½	27.65	5.18	3.98	23
	Mixed charge	10⅓	20½	30⅚	44.49	6.59	4.87	24
2	70	13	29	42	27.33	3.45	3.61	3
2	71	10½	27½	38	31.42	7.38	5.18	5
2	72	10½	17	27½	26.07	6.41	5.49	7
2	73	6	23	29	27.02	3.61	3.92	9
3	74	9	15	24	29.52	6.28	3.87	22
3	75	10	7	17	26.71	4.40	6.28	21
4	88	9	20	29	31.72	2.03	6.28	15
4	94	10⅚	19	29⅚	32.99	3.61	4.08	14
4	95	14⅓	98	112⅓	35.19	9.66	6.12	13
4	96	6⅚	12	18⅚	44.49	2.98	3.67	18
4	97	8	5½	13½	29.85	6.12	4.24	20
4	98	11½	27½	39	25.75	4.40	3.14	2
4	99	9½	16	25½	31.42	4.08	4.08	4
4	100	9	8½	17½	27.96	4.01	3.61	6
4	101	6½	14½	21	25.13	3.92	4.08	8
4	102	18¾	27½	46¼	29.21	6.91	3.77	10
	Mixed charge	6	32	38	45.25	6.59	4.87	11
	,, ,,	19½	66½	86	26.71	4.08	5.34	12

Just as soon as the tub is empty a fresh charge is put in. The twenty-four tubs are kept constantly working. Three men have entire charge of the tubs; they only work during the daytime. The leaching, however, is continuous, and is done by two Chinamen on each shift. They fill all the tubs, do all the testing, and take the probe samples. No other persons are allowed to touch the tubs while they are under their care. When the leaching is finished three labourers discharge the tubs, and carry the tails to the dump-heap. The head leacher is paid $2.50 and his helper

from \$1.00 to \$1.50 per day; 50 tons or 60 tons a day are leached by these two men.

VI. Precipitation of the Silver.

The hyposulphite liquor containing the silver is run directly from the leaching vats to the precipitating tanks, Fig. 189, which are 8 ft. in diameter and 12 ft. deep. When it is possible these tanks should be set 3 ft. below the bottom of the lixiviation tanks. When, however, the grade of the ground does not admit of this, all the tanks are placed on the same level and the transfer made by means of a Körting injector. All the solutions except the silver liquors are carried about the mill through wooden launders, carefully jointed with tar, and also painted on the inside with it. This is an excellent method. Iron pipes were formerly used, but they decomposed the solutions and sometimes precipitated the silver. Iron pipes lined with tar would be a better arrangement, especially in those mills whose work is not continuous. These might be ordinary steam-pipes, 5 in. in diameter, which are screwed together. Wooden troughs without the coating of tar were formerly used, but it was found almost impossible to keep them tight, and the penetrating nature of the hyposulphite caused a considerable loss of the reagents. Besides, when not used they shrink, which is a very great disadvantage, as they leak for some time afterwards. Troughs hollowed out of solid wood are used in some works, but they are not easily made and are more difficult to manage. The whole of this inconvenience, however, is remedied by the use of iron pipes lined with tar.

The silver liquors from all the leaching tubs are run into one precipitation vat until it is filled to within 15 in. to 20 in. of the top, the flow is then turned into the next vat. When this is done, the silver is thrown down at once, calcium sulphide being added until no precipitation takes place. The tanks for the storage of the calcium sulphide are made of $\frac{3}{16}$ in. boiler iron. They should have a capacity of from 90 to 100 cubic feet. The solution during the addition of the calcium sulphide must be agitated to make a thorough mixture, and is then allowed to settle, and the clear liquor which is reconverted into calcium and sodium hyposulphite by the calcium sulphide is drawn off into

a receiving tank on a lower level, and from there forced into a tank on the top floor, from which all the solution used runs. The reaction which takes place is

$$2\,NaAgS_2O_3 + 2\,NaCl + 10\,H_2O + CaS = Ag_2S + 2\,Na_2S_2O_3 + CaCl_2 + 10\,H_2O.$$

This solution in Mexico is at 25 deg. to 30 deg. Baumé; in the West it is much weaker.

Where various ores are treated, it requires a little time to ascertain just the quantity of sulphide to add, but when the same ores are constantly being used, it is simply measured in from a pail. There is no danger of loss, as the liquors are carefully tested. Care must be taken not to have the sulphide in excess. Only that portion of the sulphide which precipitates the silver is converted into calcium hyposulphite; any excess of sulphide would cause a precipitation of the silver already dissolved in the solution tanks as sulphide, which would not be dissolved by the hyposulphite and would consequently be lost in the tails. If there is an excess of sulphide some of the silver liquor must be added to neutralise it. At the Bertrand Mill the calcium sulphide added in the precipitation of the silver keeps the liquor up sufficiently with the weekly addition of about 50 lb. of hyposulphite. As soon as the clear liquors are run off other liquors are run in and precipitated, clarified, and so on, for about two weeks. At the end of that time the tanks must be discharged. There are three precipitating tubs for the silver sulphide and three for the precipitation of the base metals with the same sulphide when water is scarce.

A great deal of trouble has been experienced in the various works in pumping these solutions. In the Bertrand Mill an ordinary Babcock steam pump was used, but iron pumps were found not to work well, and steam pressure was then used to force the liquors up, which had a great many inconveniences. The simplest way is that formerly used at the Old Telegraph Mine, which was to use an ordinary lifting pump made with a wooden cylinder lined with hard rubber, with a hard rubber plunger, which cost less than $50. This pump worked perfectly for a long time without repairs.

Every fifteen days the silver sulphide is collected. The arrange-

ment is made to have the whole mill cleaned up at one time. To do this the clear liquor on the top is first drawn down by the rubber tube C, Fig. 189, to the end of which a float A is attached which allows the top of the tube to be just below the level of the liquor. The float A is so wide that it prevents the tube from reaching the precipitate on the bottom. The tube is then hung up so that the float occupies the position D. The sulphide on the bottom of the tanks is stirred vigorously until it is in the state of a thin mud. It is then drawn off from the bottom of the tank, through the stop-cock E, into the receptacle B, from which it is carried by launders to the filter press or, as at the Bertrand Mill, run over cloth filters. These filters consist of a series of frames about 2½ ft. square, over which sacking is securely fastened. There are thirty of these frames, three in a row, built on the same table, which is surrounded

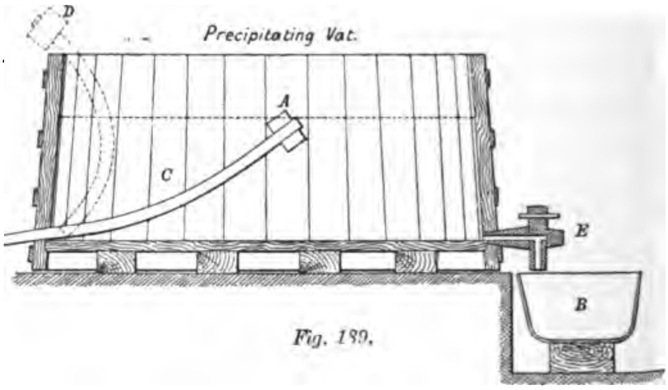


Fig. 189.

by a rim 3 in. high. Underneath the table is an inclined trough which carries all the liquors to the collecting vat. The turbid liquor from the precipitating vat is run on to this table and flows from one sacking to the other. The solution drains through and the pasty sulphide remains on the sacking. This operation is constantly repeated until the silver sulphide has accumulated in considerable quantity on the sacking. It is then washed with clear water to remove the excess of calcium sulphide until it is perfectly sweet. It is then left to dry. In drying, it cracks in every direction. It is shovelled into a car and carried to the reverberatory furnace to dry and roast. As there is a considerable

amount of water in the pasty mass, it is in some works collected in canvas bags and the water squeezed out with a press before it goes to the furnace. It would be much better and simpler to use a filter press, such as is constantly used in the metallurgical works on the Continent, to separate the silver sulphide. The silver sulphide would then be made in a few moments into cakes which could be easily roasted, but these filter pumps are but little known in the United States.

VII. Roasting the Silver Sulphide and Melting for Bullion.

The silver sulphide is now ready to be dried in a reverberatory furnace, as it still contains some water. The heat must be quite low at the commencement, only just enough being used to burn off the sulphur. It can only be increased very slowly at first for fear of melting the sulphides. It is roasted until most of the sulphur is driven off, and the heat is then raised to as high a temperature as possible without melting. When sulphur is scarce the sulphide is heated in a retort, and the sulphur condensed to be used over again. Mr. O. H. Aaron proposes to take the freshly precipitated sulphide and boil it, adding freshly slaked lime in small quantities at a time to the liquor, which is kept constantly agitated. When this is carefully done the calcium polysulphide will be formed much more quickly than by the direct action of the sulphur, and most of the sulphur in the silver sulphide will be regained. The liquor is decanted, and the residue is dried and roasted. After roasting, it is a grey mass composed almost entirely of metallic silver.

Formerly the roasted silver was melted in graphite crucibles, but this was found to be too expensive. It is now cupelled with the addition of lead in an English cupel. The litharges are rich in silver and are all reduced to form the lead for the next cupellation. From the cupels it is cast into bricks and is between 800 and 900 fine. No attempt is made to make it finer, as it is found cheaper to sell it than it is to refine it.

There are sixty men in all employed in the Bertrand Mill. The following Table gives the special occupation of each of these men and the wages they receive.

MEN REQUIRED TO RUN THE BERTRAND MILL DURING TWENTY-FOUR HOURS.

Roasters—furnace men	2 @	$4.00	$8.00
,, feeders	2 ,,	3.00	6.00
Driers and rollers	10 ,,	3.00	30.00
Cooling floor	6 ,,	3.00	18.00
Leaching floor, Chinamen ...	2 ,,	2.50	5.00
,, ,, ...	3 ,,	1.50	4.50
Rock breakers	2 ,,	3.00	6.00
Flue cleaners	4 ,,	1.50	6.00
Blacksmith...	1 ,,	6.00	6.00
,, helper	1 ,,	3.00	3.00
Carpenters...	2 ,,	5.00	10.00
Engineer	1 ,,	5.00	5.00
,, ...	1 ,,	4.00	4.00
Fireman	1 ,,	2.50	2.50
Assayer	1 ,,	5.00	5.00
Lamp cleaner	1 ,,	1.50	1.50
Foreman	1 ,,	6.00	6.00
,, ...	1 ,,	5.00	5.00
Office helper	1 ,,	1.50	1.50
General helper	1 ,,	3.00	3.00
Watchman	1 ,,	3.00	3.00
Woodmen, Chinamen	3 ,,	1.50	4.50
,, ,, ...	1 ,,	3.00	3.00
			$146.50

All the charging and discharging and leaching is done by four Chinamen and three Irishmen. The total cost of milling the ore is $6.50 per ton. The cost of mining and delivering the ore at the mill is about $2.50, which makes the total cost about $9.00 per ton of actual expense on each ton of 30 oz. ore.

It must not be taken for granted that the bullion produced from this process is necessarily purer than that produced by any other. The contrary is likely to be true. If the ore contains a large quantity of base metals, particularly lead, these would be only partially affected by amalgamation, while most of the lead contained in the ore will be likely to be found in the bullion made by leaching. The reason for this is that while all that portion of the lead which is in the form of chloride may be leached out, and, by the judicious arrangement described, the other metals can be separated, none of the lead present as sulphate will be attacked by the water, while most of it will be dissolved by the hyposulphite and will be precipitated by the

sulphide, thus using a larger amount of the reagent, and being a direct source of loss. This, however, except in very poor ores, is not a grave inconvenience. In other respects the bullion is likely to be purer. The loss is practically very small. The outlay for chemicals is also very small, as they are constantly regenerated, and the same water can be used over and over again. The amount of necessary loss in the tails will depend, other things being equal, on the price of labour. It is very easy to ascertain this by making a daily clean up, or making a time assay, to ascertain exactly how much is being extracted in a given time. The assay shows that at the Bertrand Mill 85 per cent. of the silver in the chloridised ore is saved. From 50 to 60 tons are easily treated in the twenty-four hours. The average product is from $30,000 to $40,000 per month; that of the Triumfo Works is from $50,000 to $60,000 per month.

RUSSELL'S PROCESS.*

When the ores contain any considerable quantity of base metal, especially lead, the Von Patera process cannot be used, as the hyposulphite solution acts on the lead as well as the silver. The Russell process is, however, adapted not only to such ores, but also to the treatment of the tails of the Patio and the Pan as well as those of other lixiviation processes. It consists of treating the ore by six different operations.

1. Crushing the ore and roasting it with or without salt in any of the furnaces generally used.
2. Lixiviating the roasted ore with sodium hyposulphite.
3. Treating the lixiviated ore with the extra solution of sodium and cupreous hyposulphite.
4. Precipitation of the lead by sodium carbonate.
5. Precipitation of the gold, silver, and copper.
6. Treatment of the sulphides.

For poor ores, according to their composition, the process would be somewhat modified by either modifying the roasting or not roasting at all, and if there were no chlorides present, doing away with the roasting and first lixiviation altogether, and commencing at once with the extra solution.

* Trans. Am. Inst. Min. Eng., vol. xiii., page 47; vol. xv., October, 1886; ENGINEERING, vol. xxxviii., p. 239.

I. Crushing and Roasting the Ore.

(*a*) *Crushing.*—Up to within a short time the almost universal instrument used for crushing was the stamp mill, which has attained its greatest perfection in the United States. A recent examination of nearly all the greatest dressing works of Europe shows that on the Continent it is being gradually abandoned for rolls, some of the works having abandoned it altogether, while others are only waiting their extensive repair to take the stamps out. With stamps it is not generally practicable to crush through a coarser screen than a No. ~~30~~ 20, which is equivalent to a No. ~~40~~ 30 revolving screen where the crushing is done by rolls. There is little doubt that the unintelligent use of rolls in the early days on Lake Superior had much to do with the prejudice against them in the United States. It seems to be almost certain, however, that we shall come back to them. They are already being introduced with great advantage in the West. For purposes of lixiviation there seems to be no doubt that they are superior. They not only give a pulp of more even fineness, but it contains less slimes, which for purposes of lixiviation, is important, as the liquors should pass evenly through the ore and attack all parts of it. With rolls a much closer hand-picking and sorting can be made. Too little attention is generally given to hand picking in the West, owing probably to the fact that much richer ores are treated there than in Europe. Apart, however, from the consideration of the much greater equality of the grains of ore, and the less formation of slimes, is the important consideration of cost. For the erection of rolls, the cost is not only much less, but the parts themselves are simpler and less likely to get out of order. The steel of which the outside of the roll is made can be used until there is danger of its cracking from pressure, and can then be replaced with but little cost of time and labour, so that the useful effect of the material is much greater than with stamps. The amount of repairs required, on account of the simplicity of the parts, is much less, and much less skilled labour and fewer men for the same output are required. There is also but little doubt that the actual output per horse-power of the roll is greater, as there is no necessity in most cases in order to chlorurise the ore perfectly, to have it very fine;

and as a very fine ore interferes with rapid filtration there seems to be no doubt of the superiority of rolls over stamps for crushing ore that is to be lixiviated. Rolls have been successfully used for some time at the Bertrand Mill, at the Ontario, and several other mills in Montana and in Nevada, and are being introduced elsewhere, and where very fine crushing is not required will no doubt take the place of stamps as they have done elsewhere. Even in the case of a very finely impregnated ore it is a question whether it will not be advisable to treat the ore first in rolls, and when all has been extracted from it which can be at that grade of fineness, to treat the residue with stamps as is now done in one of the largest dressing works in Europe. This, we believe, has never been done for lixiviation works, and except in the case of gold very finely disseminated in the gangue would hardly be advisable. There has been too little experience with the lixiviation process to know whether such a case would be likely to occur. In all the cases where rolls properly constructed and managed have been brought into competition with stamps, the latter have been taken out as soon as they were worn enough to require extensive repair, and rolls have taken their place.

(*b*) *Roasting the Ore.*—The problem to be solved in all the processes where it is necessary or advisable to transform the metal into a chloride by roasting it with salt has been how to perform this operation so as to give the highest percentage of chlorides. This has been especially necessary up to the present time with the lixiviation processes, since metallic silver and most of the other compounds of silver in the absence of soluble salts of copper are but slightly soluble in the hyposulphite solutions used for lixiviation. This difficulty and the impossibility of producing a bullion free from lead when it has been present in the ores, has prevented the introduction of the lixiviation process, because in amalgamation, even when the chloruration has not been perfect, metallic silver, if present, will be acted on by the mercury, and many of the silver compounds will be decomposed at its expense, so that while the loss in mercury will be somewhat greater than it otherwise would be, the loss of silver will be less. The necessity of a high chloruration of the ore has

stood in the way of the introduction of the lixiviation process, more especially as the presence of any considerable quantity of sulphurets, especially of blende, increases the difficulty and prolongs the time of roasting. It has, therefore, been the practice in many of the mills to discharge the ore in such a way that it remains hot in heaps for several hours, and thus increase the chloruration effect without prolonging the time of action in the furnace and consequent danger of decomposing some of the chloride already formed. In the investigations which I have been making during the last three or four years, I have always found that in ordinary roasting, when the temperature was very high, or the operation prolonged, the amount of the precious metals extracted was always less than when the time was limited and the temperature low. Mr. Russell has come to the same conclusion, and discovered, in addition, that the loss in silver in chlorurising roasting is always less when the proportion of salt is high. It seems, therefore, desirable to perform the roasting of the ore as quickly as possible, and to allow the ore from the furnace to remain for some time in heaps before using it. For roasting the ore most of the reverberatory and a large number of the mechanical roasters have been used. Those which have the auxiliary fireplace have been the most successful, but the Stetefeldt furnace seems to be one of the best adapted for this process, however, as the roasting may be as short as desirable, while the chlorurising action may be prolonged at pleasure. The necessity of a perfect chloruration in this process is not so great as in the other lixiviation processes which use only a simple hyposulphite solution, for should any of the metallic silver fail to be acted on by the salt, or any of the sulphide or other compounds fail to be decomposed, so that it is left in the ore, or in the pan tails, it will be dissolved in the extra solution. It has been found that in certain poor ores the use of salt may be dispensed with altogether and the ore only subjected to an oxidizing roasting and then lixiviated. Oxidized ores may, in certain cases, be treated without even roasting. The extraction is not perfect, but when the price of salt or fuel is high it may be cheaper to bear a small loss in silver rather than to pay the expense of salt and of roasting. When, however, the silver is contained in lead carbonate, the ore

must be roasted, as lead carbonate is insoluble in the lixiviating solutions.

II. Lixiviation of the Raw or Roasted Ore with Sodium Hyposulphite.

The tanks which are generally used for this purpose are 10 ft. to 15 ft. in diameter and 40 in . to 6 ft. 6 in. deep. They are made of wood painted with asphalte. It is proposed* to construct very large ones made either of Portland cement reinforced with iron or of bricks saturated while hot with coal tar, or of asphalte concrete, protecting the bottom and sides with a lattice-work of wood so as prevent injury from the falling in cf the ore or shovelling it out. The discharge pipes are made of rubber. While the solution is working they are raised and fastened to the outside of the tank, Fig. 188. When the tanks are to be discharged they are lowered into the main conduit. Pinch cocks are sometimes used on them to regulate the flow of the liquid. The main conduits are made of iron coated inside and out with asphalte. The best method of doing this is to heat them up to the melting point of asphalte and then immerse them in the liquid The storage tanks from which the solutions are drawn are provided with a coil of 1-in. lead pipe, through which steam is introduced to heat them. When there are several tanks they are connected together with a 3-in. pipe, which runs over all the tanks, and is so arranged that each tank can be supplied independently of the other. The solutions are admitted to the tanks by rubber hose.

The pulp is first treated with water; when this is admitted above on an ore which has been roasted with salt, it always contains some silver, and is allowed to flow into and out of a tank containing scrap iron, where the copper and most of the silver is precipitated. The vat is filled full of water and the discharge stopped as soon as the water above it reaches the surface of the ore. After the leaching with water has been done, the hyposulphite is turned on until the vat is full, and the leaching continued until all the solution necessary has passed over the ore. The solution should contain from 1.35 to 1.50 per cent. of the ordinary sodium hyposulphite in water to which sufficient sulphuric acid has been added to neutralise any caustic soda

* Trans. Am. Inst. Min. Eng., vol. xiii., p. 110.

it may contain. All the salts of silver except the sulphide are more or less attacked by this solution. Its object, however, is to dissolve out the chloride which is either in the ore or has been formed by roasting. When silver chloride is contained in considerable quantities in the ore associated with other minerals it is sometimes leached out, and the residue either treated or concentrated and sold. The salts of copper and lead are also dissolved. The complete solution of the lead, however, requires a concentrated solution. As the solvent power of the liquor for the base metals is increased by concentration it must be made dilute, and as lead is more soluble in hot than in cold solutions, and as any considerable quantity of lead dissolved in the hyposulphite solution decreases its solvent power, the dilute liquor is heated only to a temperature between 25 deg. and 50 deg. Cent. In this way the most of the precious metals and the least lead will be dissolved. Whenever the liquid becomes caustic it must be neutralised by the addition of sulphuric acid or copper sulphate, which neutralises the caustics, and this last salt is the neutraliser in the extra solution when the material being leached contains caustic lime.

The liquor is allowed to remain on the ore long enough to dissolve out most of the silver chloride, by which time all the other salts, except the sulphide, will have been more or less attacked, some of the gold and most of the copper dissolved. The liquor is now drawn off and the pulp is ready for the extra solution. If the ore is crushed so fine as to make the liquor pass through it slowly, it may be forced through by using a stand-pipe, an exhaust, or a force-pump. To be sure of the quickest and easiest solution, it is well to make the liquors pass through the ore several times by draining the tanks and pumping up the liquor, or by arranging some system by which it will run out of one tank into the other by gravity. Such an arrangement expedites the solution and diminishes the amount of labour.

In treating a charge of ore which has not been roasted, no washing with water is required. The vat at Bullionville is 11½ ft. in diameter, is filled 40 in. deep with dry ore, which makes it quite full; the ordinary solution is then turned on above and below at the same time. The ore immediately

begins to sink, and by the time the tub is full of solution the surface of the ore has sunk about 5 in. As soon as the tub is full the leaching upward is discontinued, and the leaching downward begins, the hose leading from the bottom of the tank being connected with a No. 0, the smallest size, Allen lead-lined syphon pump, which cost $22 each, one being attached to each tank. This pump, acting on the principle of the ejector, produces and keeps a vacuum under the filter cloth, and thus draws the solution rapidly through. At the same time it heats the solution from 90 deg. to 110 deg. F., at which point it acts most energetically. It is drawn through the 40 in. of ore at the rate of 40 to 65 cubic feet an hour, according to the percentage of slimes present in the ore. The ordinary solution is used in this way for four hours, being kept running on top of the ore. At the end of this period it is shut off for a short time, but the leaching is continued a few minutes until the solution on top of the ore has sunk to the surface of the ore. The leaching is then discontinued for a time. If the pumps used are made of wood they should be saturated with asphalte; rubber or hard lead answers the purpose better. In some cases the extra solution is used first, but it is generally preceded and followed by the ordinary solution. If the leaching is commenced with the extra solution, its volume must be sufficient to cover the charge after the circulation has commenced. The extra solution is always circulated. The volume of the ordinary solution used is from 20 cubic feet to 90 cubic feet, and the time ordinarily from twelve hours to thirty hours.

III. Leaching the Lixiviated Ore with the Extra Solution of Sodium and Cupreous Hyposulphite.

Mr. Russell has discovered that if ores of silver are leached with sodium cupreous hyposulphite, formed by adding copper sulphate to sodium hyposulphite, not only native silver, but silver sulphide and its compounds of arsenic and antimony will be dissolved. This extra solution, however, while it acts energetically on the other compounds of silver, is a much poorer solvent for the silver chloride than the ordinary hyposulphite solution. The compounds of gold are also dissolved, but the

solvent power of the solution for gold is not increased by the addition of the cupreous sulphate, while it is very much increased for silver and its compounds, except the chloride.

The material left in the tails which has not been dissolved by the previous solution may be treated by this extra solution so as to extract a considerable percentage of their precious metal contents. Ores roasted without salt, or the raw ores themselves, may also be treated, and a very considerable percentage of the precious metals contained in them be extracted. This extra solution is made by dissolving 18 parts of sodium hyposulphite and 10 parts of copper sulphate in a small quantity of water, and mixing the solutions, decanting the liquid from the precipitate, washing it, and then redissolving it in water containing from 1 to 2.5 per cent. of sodium hyposulphite. The solution so prepared may be used for several successive charges, and only turned into the precipitation tanks when the cupreous hyposulphite is nearly exhausted. It may also be made by adding so much of the saturated solution of copper sulphate to the sodium hyposulphite as would be required to make a standard solution. This solution is not permanent. A considerable portion of the sodium cupreous hyposulphite will commence to separate at once, as a yellow precipitate, very slightly soluble in water but easily soluble in sodium hyposulphite, and as it is the solution, and not the precipitate, which acts, its value will be rapidly destroyed. The more concentrated the solution, the more rapidly the precipitate will form. Hence it is much more advantageous to form the precipitate first, and then dissolve it, and in this way to form the permanent solution. The amount required to form the standard solution is 1 per cent. copper sulphate to 2.25 per cent. sodium hyposulphite, as more copper sulphate forms a precipitate, and renders the solution weaker. This precipitate dissolves, however, when more sodium hyposulphite is added. An excess of copper is, therefore, very undesirable. An excess of hyposulphite is not so much so, although the weaker the solution the less efficacious it is on native silver and silver sulphide. Dilute standard solutions at a temperature of about 50 deg. Cent. act more energetically than stronger solutions at a higher or lower temperature. Extra solutions made from calcium hyposulphite

act as energetically as those made from sodium, while the potash compound is much less powerful. When a solution containing an acid is heated to 55 deg. Cent., the cupreous hyposulphite decomposes into copper sulphide and sulphuric acid. The stock solution from which the extra solution is made should therefore not contain free acid. It never becomes caustic, as this is constantly neutralised by the cupreous sulphate. The solution is let on to the ore from above, filters through, and is pumped up again several times, in order to dissolve out as much as possible of the metals, by pumps made of hard rubber or lead.

The extra solution cannot be kept in stock, as it is liable to be affected by various causes, and in order to get over this difficulty of making it, it is made only as it is wanted during the process. To do this at Bullionville a wooden box with perforated bottom is placed just under the valve of the solution pipe over each tub. Into this box is put 3 lb. of blue stone for each ton of ore in the ore tub, and 1 lb. of hyposulphite for each ton of ore, making 37½ lb. of blue stone and 12½ lb. of hyposulphite for each charge of 12½ tons. The solution is then again allowed to run on to the ore, and in doing so it dissolves the blue stone and hyposulphite. When the cupreous hyposulphite solution stands 10 in. deep on the top of the ore in the ore tub, the solution is shut off and the leaching downward is continued. When the 10 in. of solution on top of the ore has decreased to 2 in., the discharge end of the syphon pump is connected with a hose hanging down by the side of the ore tub, and leading over the side. If the quantity of extra solution on hand falls short of the requisite quantity, ordinary solution is added to it to make up the deficiency. When a sufficient quantity of the extra solution thus weakened has passed, the hose is disconnected and the usual leaching is again commenced. The solution is thus kept rotating through the ore for four to six hours. As soon as the two inches on top of the ore has been reduced to the level of the ore, the ordinary solution is turned on again, and the leaching continued for 3½ hours. At the end of that time the solution on top of the ore is shut off and the leaching continued until the ordinary solution has sunk to the surface of the ore. The wash water is then turned on, and is continued for about four hours,

until the ore is thoroughly washed. The tap-hole in the bottom of the tub is then opened and the tailings sluiced out.

As all of the solutions must be heated to get the best effect, this is done with coils of lead pipe placed in the tanks, through which steam is made to pass, which thus keeps the solutions at the proper temperature. The quantity of silver dissolved out from the ore will depend on the surface exposed to the liquid. Hence the ore should be as fine as possible, without being so fine as to be in the shape of dust. If the roasting has been properly done, it will always be greater than the assay, as the solutions have longer time to act on the large scale than on the small. The question might be asked, Why not dispense with the first lixiviation altogether and use only the extra solution? The results of a large number of experiments have shown that in almost every case when the two solutions were used one after the other, more of the precious metals were extracted than when only one was used. It has also been found that in the treatment of some raw ores the difference in favour of the extra solution has been as high as 60 per cent., and in chlorurised ores as high as 30 per cent.

Sometimes it may be worth while to treat raw ores, especially when they contain much native silver, by the extra solution first. Thus it has at Bullionville been found most expedient to use the extra solution first and the ordinary solution afterwards. The amount of silver extracted from these ores has been $10.40 per ton. It has been found that silver may still be extracted from the tails of the amalgamation process by the extra solution, and it is probable that when ore can be amalgamated directly, lixiviation can take its place. There is, however, one class of ores which cannot be treated without previous roasting; these are those in which the silver is contained in lead carbonate.

When the ore contains no caustic lime, half the ordinary solution is added first and then the extra solution is circulated, 5 to 10 per cent. less of it being used than would be necessary to saturate the ore, in order to avoid waste. Then the ordinary solution is put on again. The time required for the ordinary solution is from twelve to thirty hours. That for the extra

solution is from four to six hours. When the ore contains considerable caustic lime immediately after it is washed with water, the leaching is suspended and the ore is impregnated with copper sulphate of the strength of 1 lb. to the ton of ore, dissolved in water enough to fill the vat to the depth of from 1 in. to 2 in. This is done to prevent the weakening of the extra solution by the water contained in the ore. It is added as soon as the wash water has sunk below the surface of the ore. A large quantity of cold extra solution is then run through first to saturate it, but without circulation. It is put on in several charges, commencing with one of .4 per cent. and gradually decreasing in strength to .1. This is done in order to saturate the caustic lime. The volume of extra solution varies from 18 to 40 cubic feet per ton. The ordinary solution is then put through, also cold. Its volume varies from 20 to 90 cubic feet.

The following experiments were made at Lake Valley.* The roasted ore while still red hot, was cooled by a spray of water. The result was a reduction of 10 per cent. in the leaching as compared with the same ore which had been allowed to become comparatively cool. In leaching the ore with water a briny liquid was obtained, which held several ounces of silver per ton of ore in solution. Two pounds of sulphuric acid per ton of ore neutralised the caustic so as to give an acid reaction. By heating the solution, the silver was precipitated with iron in twelve hours. When the ore after leaching was treated, first with the ordinary and then with the extra solution, no silver was dissolved by the extra solution, but when after leaching with water the extra solution was used, the extraction of the silver increased from 20 to 25 per cent. over what was extracted by the usual method. The cupreous hyposulphite neutralised the calcium hydroxide and prevented the solution from becoming caustic. Both the ordinary and the extra solution should be used fresh, hence the effort is made when it can be done to form it as it is used. If kept unused for any length of time, sulphates are formed by the action of the air, and the solution loses its solvent power.

It appears as the result of trial, that part of the silver in the

* Trans. Am. Inst. Min. Eng. vol. xv., May, 1886.

roasted ore exists as sulphate. It has been proposed to use for such products as are now treated by the Ziervogel process, a much less careful and consequently a cheaper roasting for silver sulphate,* and then to add the extra solution to the ore, after it has been leached with hot water without removing it from the tanks. All the silver in the ore, except such as was contained in any fused or fritted product, which ought not to be there at all if the temperature has been properly regulated, would be dissolved. This would make the complicated processes now used on the Ziervogel tub residues unnecessary, and would probably be much more satisfactory in its results. The Table on next page shows the effect of leaching by the two solutions.

The solution from the tanks may contain arsenic, antimony, lead, copper, silver, gold, and lime. No iron manganese or zinc can be present if the ore has been properly washed. Arsenic and antimony come from the arsenic and antimony compounds in the roasted ores. Lead comes from the lead sulphide in both raw and roasted ores. Copper comes from the extra solution, from chlorides in roasted ores, and from carbonates in raw ores. Lime comes from calcium sulphate and caustic lime in roasted ores, and from gypsum in raw ores. The amount of lead that is contained in the solution rarely exceeds 5 lb. to the ton of ore. Lead sulphate is more easily soluble in warm than in cold solutions, and in concentrate than in dilute solutions, but practically with such small amounts these circumstances make but little difference. A large amount of lead sulphate in the ore does not practically affect the results, as all of it is not dissolved. While it is desirable to reduce the quantity of lead, it is to be borne in mind that the only object of the process is the extraction of the silver, and that the arrangement of details must be made with reference to it. The total amount of base metals is only a few pounds to the ton of ore. It is removed at this stage of the process, because if allowed to remain it would deteriorate the quality of the bullion made from the precipitated sulphides.

IV. Precipitation of the Lead by Sodium Carbonate.

Mr. Russell has discovered that lead carbonates are not

* Trans. Am. Inst. Min. Eng., vol. xiii., p. 80.

Leaching Tests made at Leadville.	Ounces of Silver per Ton of Ore.	Hyposulphite Solution alone.				Russell's Process.			
		Raw Ore.		Roasted.		Raw Ore.		Roasted.	
		Ounces Silver Leached.	Per Cent. Leached.	Ounces Silver Leached.	Per Cent. Leached.	Ounces Silver Leached.	Per Cent. Leached.	Ounces Silver Leached.	Per Cent. Leached.
No. 1 contains no lead, is highly silicious, most of the silver is present as chloro-bromide	42.0	37.5	89.2	37.8	90.0	39.5	94.0	39.0	93.0
No. 2, basic ore, 20 per cent. manganese, and very little lead	18.0	12.0	67.7	14.5	80.0	12.0	67.7	15.2	84.4
No. 3, partly oxidised iron ore, 30 per cent. lead chiefly as galena	12.0	7.0	58.3	10.1	84.1	10.0	83.3	11.2	93.3
No. 4, highly manganiferous ore with considerable lime	14.0	9.0	64.2	9.2	65.7	9.1	65.0	9.4	67.1
No. 5, pure sulphuret ore, 10 per cent. lead, 30 per cent. zinc	14.5	6.0	41.3	11.3	78.6	7.5	51.6	12.1	83.4
No. 6, ore from Sugar Loaf Mountain; very silicious, silver as sulphuret	38.0	27.0	71.0	35.0	91.0	29.5	77.6	36.8	97.3
No. 7, flue dust from smelter, 26 per cent. lead	15.0	8.0	53.6	12.0	79.6	7.8	52.0	12.5	83.3

soluble in the hyposulphite solution, while other salts of lead, silver, and copper are. If a solution of pure lead sulphate in sodium hyposulphite is treated with sodium carbonate, the precipitation is so complete that sulphuretted hydrogen gives no precipitate in the filtrate. If sodium carbonate, therefore, is added to the solutions drawn from the leaching tanks the lead will be precipitated, while the other metals will remain in solution. The solution is made with the ordinary stock solution of hyposulphite by adding 12 lb. to 16 lb. to each cubic foot of stock solution. This is done in the storage tank. The separation of the lead is so complete that no trace of it remains. The calcium, iron, manganese, and zinc carbonates are insoluble. Any soluble salts of these metals, which were contained in the ore or that were formed in the first stages of roasting, would be decomposed in the last stages of the roasting, or if they were not they would be washed out in the first wash of water, and would not be found in the pulp. To effect this separation of the lead it is most convenient, as well as economical, to make use of the ordinary commercial soda ash, which can be procured at a low price almost anywhere. Only such as contains from 70 to 76 per cent. should be used. This substance is, however, never pure. It is likely to contain besides the sodium carbonate both caustic soda and sodium sulphide, which, if present in the solution, would precipitate the silver in the lead. A solution made from such impure material will throw down some silver as sulphide and produce a lead carbonate which may contain as much as 100 oz. of silver to the ton. To prevent this, the solution of soda ash is first dissolved in water containing 1.50 per cent. of sodium hyposulphite, then boiled with sulphur, which forms sodium hyposulphite and polysulphide, with the caustic soda. The polysulphides are then precipitated with copper sulphate. Copper sulphide is formed, and this is insoluble in the hyposulphite solution. This separation of lead makes the use of sodium hyposulphite absolutely necessary. Calcium hyposulphite cannot be used because the calcium would be precipitated with the lead. Soda ash is now made at Syracuse, New York, by the Solway process 98.7 per cent. pure, and costs there 1.4 cents per pound. As it is entirely free from caustic

soda and sodium sulphide, it can be used as it is without any purification.

To effect the separation of the lead the liquors are drawn off into separate tanks and the necessary quantity of the purified sodium carbonate added to the solution. The solution is usually stirred by hand with a paddle of hard wood. Mechanical agitators should be used to make the mixture perfect; these can be arranged so as to be worked with but little expense. The lead carbonate settles very rapidly, and in less than an hour the clear supernatant liquor can be decanted into the precipitating tanks. The precipitate is allowed to accumulate for several charges before it is removed or sent to the filter press. When sufficient lead carbonate has accumulated it is either pumped through the filter press or collected by hand, dried, and sold. The lead carbonate is very pure. There are no base metals in it. If, however, any lime was in the solution, it comes down with it. It contains only ½ oz. to ¾ oz. of silver to the ton. It is always wise to precipitate the lead. It may be even a source of small profit when the amount from its sale is greater than the cost of the soda. Where it has no value the gain is still great, as the lead would have to be precipitated by sodium sulphide, a more costly reagent, but the net gain is in having the silver and copper sulphides free from lead. Only the quantity necessary to precipitate the lead should be added, for while it does not injure the hyposulphite solution, the impurities it might contain, such as caustic soda, would have to be neutralised, which would increase the consumption of sulphuric acid, as the acid would act upon the carbonates as well. In practice less than 1 lb. of soda is used in precipitating 1 lb. of lead in the ore.

V. Precipitation of the Precious Metals.

The copper, silver, and gold are precipitated from the hyposulphite solution by sodium sulphide; this is necessary, because any calcium entering the solution would be precipitated with the lead of the sodium carbonate. The sodium sulphide is prepared by boiling sodium hydrate with sulphur, and as the hydrate is readily soluble, the formation of the sulphide is much more easily effected, and the solution itself is much more effective

as a precipitant than the calcium sulphide, which, however, in any case could not be used in the presence of lead, whose hydrate is not so soluble. The sodium hyposulphite, formed by the precipitation, can be heated to 100 deg. without much detriment, as it decomposes slowly, while the calcium hyposulphite so produced can only be heated to a low point without decomposing. The only objection to its use is the cost of the sodium hydrate as compared with the caustic lime. Sodium hydrate comes in sheet-iron drums, holding from 630 lb. to 700 lb., and costs in New York about 2½ cents per pound for a 60 per cent. article. Once opened it should not be left exposed, as it absorbs carbonic acid from the air. Much less sulphur is also used in making the sodium sulphide, and this with the other advantages it possesses as a precipitant makes it better and usually as cheap, if not cheaper, than the calcium sulphide. A practical experiment to determine this was made at the Cusihuiriachic Mill in Mexico, which treats 50 tons in twenty-four hours.* The mill was run thirty-eight days on sodium sulphide and twenty-one days on calcium sulphide with the following results:

	Sodium Sulphide 38 Days. cents.	Calcium Sulphide 21 Days. cents.
Average value of the ore treated, in ounces	35.1	39
4.4 lb. of caustic soda per ton of ore at 10 cts.	44.	
2.9 ,, sulphur ,, ,, 7 ,,	20.3	
24 ,, caustic lime ,, ,, 1 ,,	...	24
10.3 ,, sulphur ,, ,, 7 ,,	...	72.1
Total	64.3	96.1

If the cost of the 39 oz. of ore treated with calcium sulphide is reduced to 35 oz., we have:

	cents.
21.6 lb. of caustic lime at 1 cent	21.6
9.3 ,, sulphur at 7 cents	65.1
Total	86.7

A difference of 22.4 cents, or, on 50 tons, of $11.20 per day.

The price of 74 per cent. caustic soda in New York is 3 cents per pound; that of pulverised sulphur 2½ cents. In this market there would therefore be a saving of 2.8 cents per pound of ore of a similar character. Calcium sulphide can only be cheaper

* "Engineering and Mining Journal," vol. xlii., p. 383.

than sodium sulphide when the lime is counted as almost nothing, and sodium sulphide is very much cheaper than caustic soda. Independent of this is the labour of making the two reagents. At Cusihuiriachic four iron tanks, 2½ ft. by 3 ft. by 6 ft., had to be kept boiling by day and night in order to supply the calcium sulphide for the 50 tons of ore to be treated, while in one half-hour enough sodium sulphide was made for 150 tons. The solution when prepared contains considerable sodium hyposulphide, and as this salt is formed from the sulphide by the precipitation of the metals, it grows stronger by use. The solutions should be used at once, as after a time the dilute solutions lose their power. While the sodium sulphide is being added, the liquor should be very vigorously stirred with a paddle. Only just the quantity of sulphide required should be used. If more has been used than is necessary, more of the silver solution should be added. The precipitating tanks should therefore never be filled to the top, for the reason that it may be necessary to add new solution, but if any silver happens to be left in the solution it is not lost, as all the solutions are used over again. In case the ore contains no lead, the solution may contain lime, which, if not separated, may come down with the silver and make it impure. It is therefore best to add the sodium carbonate, decant the liquor, and then add the sodium sulphide. The amount of sodium sulphide used depends principally on the amount of copper contained in the solution.

VI. Treatment of the Precipitates.

After every precipitation the supernatant liquor is drawn off to be used over again and fresh liquor containing metals in solution is let into the tanks; when a sufficient number of precipitations have been made, the sulphides are withdrawn and dried, as at the Bertrand Mill,* or better, are put through a filter press. If they are to be roasted the sulphides must be first dried and then carefully oxidized, so as to leave sulphur enough to form a matte with the copper, which will be very rich in silver. Or they may be dissolved without previous drying in sulphuric acid at 66 deg. Baumé to which sodium nitrate has been added.†

* See page 512.

† Trans. Am. Inst. Min. Eng., vol. xiii., page 104.

Some silver chloride will be formed from the impurities contained in the reagents, and this with the gold will remain undissolved. The sulphur which collects is used in the formation of the sodium sulphide. The escaping nitric acid is caught in a coke tower. The silver sulphate is reduced with copper to metallic silver and melted. The copper sulphate is crystallised, a part of it is used in the extra solution, and the rest either sold or precipitated with iron. The residue of gold and silver chloride is allowed to accumulate and is treated by itself when there is enough of it. When the sulphides are to be sold they should be washed before shipment in order to drive off the free sulphur they contain. The smelting works near New York pay 99 per cent. of the New York value of the silver contained in them. The price paid for gold, silver, and copper in the roasted sulphides is as follows:

For gold, $18 per ounce.
,, silver, 98 per cent. of New York quotations.
,, copper, 3½ cents per pound.
Less $100 per ton for treatment.

Russell's process is now in use in Silver Reef, Bullionville, and the Ontario Mill. At the former place it is used on tailings at the rate of 25 tons per day, the tailings assaying 6½ oz. to 9½ oz. of silver but no gold. The total expenses per ton are $1.65. The percentage of precious metal extracted averages 55 to 60. The quantity increases as the men become better acquainted with the process. The amount of blue stone used per ton is 2½ lb. This quantity gives as good results as more would do. The presence of copper carbonate in the ore, which is the only compound of copper, so far as is yet known, except the sulphate, which occurs in nature and is soluble in the ordinary hyposulphite solution, causes the sulphides to have a low percentage of silver and a high percentage of copper. The copper carbonate occurs in considerable quantity in the White Reef, but very little is found in the other reefs of Silver Reef. The former, however, is the one being worked by Russell's process. On account of the presence of the copper carbonate they contain only 3500 oz. per ton in silver. The *net* price obtained per ounce in Salt Lake for these sulphides in 1885 was 95 per cent.

of New York quotations. The plant at Silver Reef is now being enlarged and rolls added so as to treat the ore of all the reefs. With pure ores the product of the sulphide will probably rise to 12,000 oz. to 15,000 oz. of silver to the ton.

At Silver City 30 tons of tails from the amalgamation works have been treated daily, which have been concentrated since they passed the pans. They assay 30 oz. of silver and are free from gold. From these tails the Von Patera process extracts 38 per cent., and the Russell process 72.4 per cent. It would have been possible to have extracted 22 per cent. by amalgamation before leaching, but further mechanical concentration has no effect upon them whatever. The work with this process shows that it extracts within .15 of an ounce as much as the assay. When the process was first introduced three tubs were used for leaching 30 tons a day, which are 12 ft. in diameter by 40 in. deep. Each tub is charged within 2 in. of the top, the charge being 15 to 16 tons. The amount of extra solution used is equal to 6½ cubic feet for each ton of ore. This solution is made by dissolving 4 lb. of blue stone per ton of ore in the ordinary 2½ per cent. hyposulphite solution. The extra solution, therefore, contains one per cent. of blue stone and 2½ per cent. of hyposulphite. This is heated by means of a coil in the solution tank up to 100 deg. to 120 deg. Fahr. This is circulated in the tails for six hours, at the end of which time it is drawn into the precipitating tubs, and the ordinary hyposulphite solution turned on and continued for 12 hours, but is not circulated. At the end of the 12 hours the tails are washed with water for 20 hours. The tubs are emptied by sluicing, which takes 1½ hours. They are filled from a bin which holds 55 tons, which takes three-quarters of an hour. The amount of extra solution used is thus 6½ cubic feet, the amount of ordinary solution 13 cubic feet, and the amount of wash water 23 cubic feet per ton. The rate of leaching depends upon the fineness of the ore or of the tails. The average fineness is such that 90 per cent. of it passes through an 80 screen. This can be leached at the rate of only 2 in. per hour. When the fineness rises to 97, only three-quarters of an inch per hour can be leached. If it was 86, 2½ in. per hour. These depths are gauged from the way the liquid on top of the ore sinks. It would be impossible, owing to the fineness and slaty character of these tails, to leach

them at all without the vacuum, which here is not very perfect, the best that they have been able to get corresponding only to 14 in. of mercury. The use of these vacuum pumps makes the position of the tubs immaterial, so that the works can be laid out according to any plan that seems judicious, as gravity need not have anything to do with the transfer of the various solutions. This is a great advantage in certain districts where the larger part of the ground is level. For these works, as now constructed, there are three storage tanks, two for ordinary solution and one for extra solution. These are 8½ ft. in diameter by 8½ ft. high, and are made with 3 in. staves. There are three precipitation tubs of the same size. There are seven ore tubs 14 ft. in diameter and 5 ft. high, which hold from 30 tons to 35 tons of tails, in which the charge will be from 4 ft. to 4½ ft. deep. The average weight of a cubic foot of these tails in the tubs dried is 105 lb. With this plant the mill can treat 100 tons to 125 tons per day. The ore is dried, conveyed, crushed by rolls, elevated, leached, and the bullion produced, at a cost of $1.80 per ton. This includes the incidental expenses of assaying, management, office expenses, &c. The sulphides resulting from the precipitation by sodium sulphide are put through a filter press with a pressure of 150 lb. to the square inch. They still contain, however, 38 per cent. of moisture. When dried, they assay about 4500 oz. per ton. After driving off the sulphur they lose 47 per cent. and assay about $9000, or 30 per cent. of silver. The percentage of copper contained is 33 per cent. When melted in the bars the bullion is about 400 fine, the impurities being mostly copper. The caustic soda which is used comes in drums, which weigh about 700 lb., and cost at Silver Reef 5 cents per pound. The drum is broken up into 10 lb. pieces and put into an iron tank and its own weight of water put over it. It is then heated with steam, and in about an hour half of the mass is dissolved, and at a temperature of about 235 deg. Fahr. Sulphur equal to two-thirds of the weight of the caustic soda is then added. This causes violent ebullition, the temperature rising immediately to 280 deg. Fahr. In about half an hour after the sulphur is added the sodium sulphide is ready for use. The total bulk is equal to an amount of water weighing three times as much as the caustic soda. The sodium sulphide will precipitate volume for volume

over one hundred times as much silver as the strongest calcium sulphide solution made in practice by steam, and seventeen times as much as the strongest calcium sulphide which can be made in the laboratory. Each pound of caustic soda in this solution will precipitate 2.24 lb. of silver, and each pound of sulphur precipitate 3.36 lb. of silver. The strongest calcium sulphide solution which can be made in the laboratory will precipitate only 1.65 lb. of silver, and the sulphur the same. When copper is to be precipitated, however, the amount of metal precipitated is much less; 700 lb. of caustic soda and 466 lb. of sulphur is sufficient for 270 tons of ore, and will precipitate about 354 lb. of metal, of which 168 lb. is silver and 186 lb. copper. The following is a Table of the cost of expenses of leaching 100 tons of raw tails per day and hauling it by team at Silver City:

RAW LEACHING 100 TONS TAILINGS PER DAY AT SILVER CITY.

	Per Day.		Per Ton.	
Hauling by team	@ $0.40	=	$40.00	=44 cts.
2 weighers...	,, 2.00	=	4.00	
			$44.00	

DRYING, CRUSHING, AND CONVEYING.

	Per Day.		Per Ton.	
1 engineer...	@ $3.50	=	$3.50	=48 cts.
1 blacksmith	,, 4.00	=	4.00	
1 helper	,, 2.00	=	2.00	
4 feeding dryer	,, 2.00	=	8.00	
2 firing	,, 3.00	=	6.00	
3 on conveyers	,, 1.75	=	5.25	
2 ,, rolls	,, 2.50	=	5.00	
1 ,, wood...	,, 2.00	=	2.00	
1 roustabout	,, 1.75	=	1.75	
3 cords wood	,, 3.50	=	10.50	
			48.00	

MILLING, LABOUR, WOOD, &c.

	Per Day.		Per Ton.	
1 engineer...	@ $3.50	=	$3.50	47¼=cts.
1 carpenter	,, 3.00	=	3.00	
2 on tubs	,, 3.00	=	6.00	
2 helpers	,, 2.00	=	4.00	
2 on precipitation tubs	,, 3.00	=	6.00	
1 ,, sulphides	,, 2.50	=	2.50	
2 roustabouts	,, 1.75	=	3.50	
2½ cords wood	,, 3.50	=	8.75	
Incidentals, oil, repair materials, &c....	,, 10.00	=	10.00	
			$47.25	

CHEMICALS.

	Per Day.	Per Ton.	
4 lb. blue stone, per ton of ore	@ 6 cts. =	$24.00	
3 „ hyposulphite „	„ 3½ „ =	10.50	
1¾ „ sulphur „	„ 4 „ =	7.00	
2½ „ caustic „	„ 5 „ =	12.50	=55 cts.
¼ „ sulphuric acid „	„ 4 „ =	1.00	
		55.00	

GENERAL EXPENSES.

Office management, assaying	$26.00	=26 cts.

Resumé of Expenses of Raw Leaching 100 *Tons per Day.*

Drying, crushing and conveying	$48.00
Milling, labour and wood	47.25
Chemicals...	55.00
General expenses	26.00
	176.25

At the Ontario Mill about 17,000 oz. of silver was produced up to June, 1885, by the Russell process, the value per ton of the raw sulphide varying from 8000 oz. to 12,000 oz. of silver. When roasted they lose about 40 per cent. of their weight, and rise in value from 13,000 oz. to 20,000 oz. By using sodium sulphide instead of calcium sulphide a product of much higher grade is obtained, as if calcium sulphide is used, the calcium unites with any sulphuric acid or sulphate of soda produced by the decomposition of the hyposulphite, or in other ways, making sulphate of lime which is insoluble and is precipitated with the sulphide of silver, and debases it.

This process has also been in use since January, 1885, at Bullionville, Nevada, where about 300,000 tons of tailings have accumulated. The capacity of the mill is about 175 tons of raw tailings per day. The cost of altering the old 10-stamp amalgamating mill into raw leaching works of the above capacity was about $8000, and the cost of a Hallidie wire ropeway which hauls 200 tons of tailings per day up to the mill, and is 3000 ft. long, was about $3700. Some details of the raw leaching at this mill are given below :*

Average weight of cubic foot of tailings, dry... ...	86 lb.
„ percentage of moisture	8 per cent.
Total capacity of tubs (level full) in cubic feet ...	323
„ weight of charge of dry ore per tub	12½ tons

* For these details I am indebted to Mr. E. H. Russell.

Average time of filling tubs with ore	$2\frac{1}{4}$ hours
,, ,, leaching with ordinary solution ...	$7\frac{1}{2}$,,
,, ,, ,, Russell's process ...	$3\frac{1}{2}$,,
,, ,, ,, water...	4 ,,
,, ,, sluicing and tailings...	$\frac{3}{4}$,,
Total time between charges	18 ,,
Average depth of charge of dry ore	40 in.
,, ,, ,, after leaching	33 ,,
Diameter of ore tubs	$11\frac{3}{4}$ ft.
Depth of ore tubs inside above filter...	40 in.
Thickness of staves and bottom	3 ,,
Total amount of solution on hand for 175 tons per day...	1550 cubic feet
Average strength of solution in hypo-sulphite ...	.9 per cent.
,, temperature	90 deg. Fahr. to 110 deg. Fahr.
Number of cakes of sulphides made by Johnson's filter press	24
Weight of small cakes, each	$8\frac{1}{2}$ lb.
,, large ,, ,, ...	24 ,,
Total weight of full charge (6 large, 18 small) ...	288 ,,
Amount of washed water required per ton of ore ...	13 cubic feet
,, ordinary solution required per ton of ore	30 ,, ,,
,, extra solution	$6\frac{1}{2}$,, ,,

A plant to treat 50 tons to 100 tons of ore per day should have:

2 hyposulphite storage tanks.
1 extra solution vat.
2 sumps for collecting the leach water and the silver sulphide.
1 tank for making sodium sulphide.
2 tanks for storing sodium sulphide.
1 tank for preparing the soda ash.
2 storage tanks for soda solution.
1 filter press.
1 solution pump.
1 Körting's injector.
2 to 3 tanks for precipitating lead.
2 to 4 tanks for precipitating sulphide.
Tanks necessary for separating copper and silver from the wash water.

WORKING STATISTICS FOR CHLORURISING AND LIXIVIATING 100 TONS A DAY OF ORE.

	Value 75-100 oz.	Value 10-30 oz.
Amount of first wash water, per ton	80 cu. ft.	40 cu. ft.
,, ordinary solution, per ton	132 ,,	45 ,,
,, extra solution, per ton	10 ,,	10 ,,
,, second wash water, per ton ...	11 ,,	11 ,,
Per cent. of hyposulphite in ordinary solution	$1\frac{1}{2}$,,	$1\frac{1}{4}$,,
,, ,, extra solution ...	$1\frac{8}{10}$,,	$1\frac{6}{10}$,,
,, blue stone in extra solution ...	$\frac{8}{10}$,,	$\frac{7}{10}$,,

Time of first wash water, per charge	9 hours.	$4\frac{1}{2}$ hours
,, ordinary solution, per charge ...	23 ,,	8 ,,
,, extra solution, per charge	5 ,,	5 ,,
,, second wash water, per charge ...	3 ,,	3 ,,
,, charging	1 ,,	1 ,,
,, discharging	2 ,,	2 ,,
Total time between charges	43 ,,	$23\frac{1}{2}$,,
Value of product per ton ...	16,000 to 18,000 oz.	9000 to 12,000 oz.
Fineness of bullion, produced	700 to 875	600 to 700

The ore is assumed to be very base.

Working Statistics for Raw Lixiviation of 100 Tons a Day of Ores.

	Value 50-75 oz.	Value 10-25 oz.
Amount of ordinary solution, per ton	80 cu. ft.	35 cu. ft.
,, extra solution, per ton	9 ,,	9 ,,
,, wash water, per ton	10 ,,	10 ,,
Per cent. of hyposulphite in ordinary solution	$1\frac{1}{2}$,,	$1\frac{4}{10}$,,
,, ,, extra solution ...	$1\frac{3}{4}$,,	$1\frac{6}{10}$,,
,, blue stone in extra solution ...	$\frac{1}{10}$,,	$\frac{6}{10}$,,
Time of ordinary solution, per charge	20 hours.	9 hours.
,, extra solution, per charge	5 ,,	5 ,,
,, wash water, per charge	3 ,,	3 ,,
,, charging	1 ,,	1 ,,
,, discharging	2 ,,	2 ,,
Total time between charges	31 ,,	20 ,,
Value of product, per ton ...	13,000 to 14,500 oz.	9000 to 10,500 oz.
Fineness of bullion produced	725 to 775	600 to 650

Working Statistics for Raw Lixiviation of 100 Tons a Day of Tailings.

	Value 8-15 oz.
Amount of ordinary solution, per ton	30 cu. ft.
,, extra solution, per ton...	9 ,,
,, wash water, per ton	10 ,,
Per cent. of hyposulphite in ordinary solution	$1\frac{1}{4}$,,
,, ,, extra solution	$1\frac{6}{10}$,,
,, blue stone in extra solution...	$\frac{1}{10}$,,
Time of ordinary solution, per charge...	Varies with the fineness of the material.
,, extra solution, per charge	
,, wash water, per charge	
,, charging	
,, discharging	
Total time between charges	
Value of product, per ton	5000 oz. to 9000 oz.
Fineness of bullion produced	350 to 600

Working Statistics of Raw Lixiviation and Concentration of 100 Tons a Day.

	Value 40-60 oz.	Value 10-25 oz.
Amount of ordinary solution, per ton	70 cu. ft.	30 cu. ft.

	Value 60-50 oz.	Value 10-25 oz.
Amount of extra solution, per ton	9 ,,	9 ,,
,, wash water, per ton...	10 ,,	10 ,,
Per cent. of hyposulphite in ordinary solution	$1\frac{1}{2}$,,	$1\frac{1}{4}$,,
,, ,, extra solution ...	$1\frac{3}{4}$,,	$1\frac{6}{10}$,,
,, blue stone in extra solution ...	$\frac{7}{10}$,,	$\frac{5}{10}$,,
Time of ordinary solution, per charge	18 hours.	8 hours.
,, extra solution, per charge	5 ,,	5 ,,
,, wash water, per charge	3 ,,	3 ,,
,, charging	1 ,,	1 ,,
,, discharging	2 ,,	2 ,,
Total time between charges	29 ,,	19 ,,
Value of product, per ton	11,000 to 12,500 oz.	6000 to 9750 oz.
Fineness of bullion produced	625 to 700	375 to 625
Value of concentrations in silver, per ton / ,, ,, ,, lead ,,	Very variable	Very variable.

Cost of Raw Lixiviation, by Russell Process, at Bullionville, Nevada, for 50 Tons per Day.

Hauling by Hallidie Ropeway (Up Grade) Daytime only.

		Per Day.	
1 man filling buckets	@ $2.50	= $2.50	=37 cts. per ton.
1 ,, dumping buckets	,, 2.50	= 2.50	
1 blacksmith	,, 4.00	= 4.00	
1 man and team	,, 4.00	= 4.00	
$1\frac{1}{2}$ cords wood, and hauling same ...	,, 4.90	= 7.35	
Repair material	,, 1.50	= 1.50	
		$21.85	

Milling Labour (Day Time only).

		Per Day.	
2 men on ore tubs	@ $3.00	= $6.00	=57 cts. per ton.
1 man on precipitating tubs	,, 3.00	= 3.00	
1 ,, press and roaster	,, 2.50	= 2.50	
1 engineer	,, 4.00	= 4.00	
1 watchman	,, 2.50	= 2.50	
1 carpenter	,, 3.50	= 3.50	
1 man on tailing pit	,, 2.00	= 2.00	
1 assayer	,, 3.00	= 3.00	
1 roustabout	,, 2.00	= 2.00	
		$28.50	

Chemicals and Supplies.

		Per Day.	
$2\frac{1}{2}$ lb. hyposulphite, per ton	@ $.06$\frac{1}{2}$	= $8.13	=$78\frac{1}{2}$ cts. per ton.
3 ,, blue stone ,, ...	,, .07$\frac{1}{4}$	= 10.88	
3 ,, sulphur ,, ...	,, .01$\frac{1}{4}$	= 1.87	
2 ,, lime ,, ...	,, .01	= 1.00	
Coke, crucibles, oils, and repairs ...		= 7.50	
2 cords wood, per day	,, 4.90	= 9.80	
		$39.18	

General Expense.

	Per Day.
Office, $12.00; management, 10.00	$22.00
Insurance and taxes	3.50
Freight on sulphides	1.50
Travelling...	2.00
	$29.00

= 58 cts. per ton.

Resumé of the Expense of Raw Leaching.

	Total.	Per Ton.
Hauling, &c.	$21.85	$0.437
Milling	28.50	0.57
Chemicals and supplies	39.18	0.785
General expenses	29.00	0.58
	$118.53	$2.372

COST OF RAW LEACHING AT BULLIONVILLE, NEVADA, FOR 100 TONS PER DAY.

Hauling by Hallidie Ropeway (Up Grade), Daytime only.

		Per Day.
1 man filling buckets	@ $3.00 =	$3.00
1 ,, dumping buckets	,, 2.50 =	2.50
1 blacksmith	,, 4.00 =	4.00
1 engineer	,, 4.00 =	4.00
1 man and team	,, 4.00 =	4.00
2 cords wood and hauling same	,, 4.90 =	9.80
Repair material	,, 1.50 =	1.50
		$28.80

= 288 cts. per ton.

Milling Labour.

		Per Day.
4 men on ore tubs	@ $3.00 =	$12.00
2 ,, cars	,, 2.50 =	5.00
2 ,, precipitating tubs	,, 3.00 =	6.00
1 man on press and roaster	,, 3.00 =	3.00
1 engineer and night foreman	,, 5.00 =	5.00
1 carpenter	,, 3.50 =	3.50
1 man on tailing pit	,, 2.00 =	2.00
1 assayer	,, 3.00 =	3.00
1 roustabout	,, 2.00 =	2.00
		$41.50

= 41½ cts. per ton.

Chemicals and Supplies

		Per Day.
2 lb. hyposulphite, per ton	@ .06½ =	$13.00
3 ,, blue stone ,, ...	,, .07¼ =	21.75
3 ,, sulphur ,, ...	,, .01¼ =	3.75
2 ,, lime ,, ...	,, .01 =	2.00
Coke, crucibles, oils and repairs	,, =	10.00
2¼ cords wood, per day	,, $4.90 =	11.00
		$61.52

= 61½ cts. per ton.

General Expense.

	Per Day.	
Office, $12.00 ; management, $10.00...	$22.00	
Insurance and taxes	3.50	
Freight on sulphides	1.75	=29¼ cts.
Travelling	2.00	per ton.
	$29.25	

Resumé of Expenses for Raw Leaching.

	Total.	Per Ton.
100 tons	$28.80	$0.288
Hauling, &c.	41.50	0.415
Milling	61.52	0.6152
Chemicals and supplies	29.25	0.2925
	$161.07	$1.6107

COST OF RAW LEACHING AT BULLIONVILLE, NEVADA, FOR 175 TONS PER DAY.

Hauling by Hallidie Ropeway (Up Grade).

			Per Day.	
4 men filling buckets	@ $2.50	=	$10.00	
2 ,, dumping buckets	,, 2.50	=	5.00	
1 blacksmith	,, 4.00	=	4.00	
1 engineer...	,, 4.00	=	4.00	=27¼ cts.
2 teams and men	,, 4.00	=	8.00	per ton.
3 cords of wood @ 4.50 and hauling ...	,, .40	=	14.70	
Repair materials	,, 2.00	=	2.00	
			$47.70	

Milling Labour.

			Per Day.	
4 men on ore tubs	@ $3.00	=	$12.00	
2 ,, cars	,, 2.50	=	5.00	
2 ,, precipitating tubs	,, 3.00	=	6.00	
1 man on press and roaster	,, 3.00	=	3.00	
1 engineer and night foreman	,, 5.00	=	5.00	=24 cts.
1 carpenter	,, 3.50	=	3.50	per ton.
1 man on tailing pit	,, 2.50	=	2.50	
1 assayer	,, 3.00	=	3.00	
1 roustabout	,, 2.00	=	2.00	
			$42.00	

Chemicals and Supplies,

			Per Day.	
1½ lb. hyposulphite, per ton	@ .06½	=	$17.06	
3 ,, blue stone ,, ...	,, .07¼	=	38.06	
3 ,, sulphur ,, ...	,, .01¼	=	6.56	
2 ,, lime ,, ...	,, .01	=	3.50	=51½ cts.
Coke, crucibles, oils, and repairs ...	,,	=	10.00	per ton
3 cords wood, per day ,, ...	,, $4.90	=	14.70	
			$89.88	

General Expense.

Office, $12.00; management, $10.00...	$22.00	=17½ cts. per ton.
Insurance and taxes	3.50	
Freight on sulphides	2.50	
Travelling expenses	2.00	
	$30.00	

Resumé of Expenses of Raw Leaching for 175 tons a day.

	Total.	Per Ton.
Hauling, &c.	$47.70	$0.2752
Milling	42.00	0.24
Chemicals...	89.98	0.515
General expenses	30.00	0.1725
	$209.68	$1.200

EXPENSE OF CHLORURISING AT BULLIONVILLE, NEVADA, FOR 100 TONS PER DAY.

LABOUR.

Ropeway.

1 team and teamster	@ $4.00	=	$8.00	
1 filling and 1 dumping bucket	,, 3.00	=	6.00	
3 men				$14.00

Rolls, Dryers, and Stetefeldt Furnace.

6 dryers (8 hours)	@ $2.50	=	$15.00	
3 firemen ,,	,, 3.00	=	9.00	
9 cooling floor (8 hours)	,, 3.00	=	27.00	
2 on rolls...	,, 3.00	=	6.00	
2 on salt ...	,, 2.50	=	5.00	
22 men				$62.00

Leaching.

4 on ore tubs	@ $3.00	=	$12.00	
2 helpers	,, 2.50	=	5.00	
2 precipitating and 1 sulphide press ...	,, 3.00	=	9.00	
9 men				$26.00

Miscellaneous.

1 superintendent	@ $10.00	=	$10.00	
1 day foreman	,, 5.00	=	5.00	
1 night ,,	,, 4.00	=	4.00	
2 engineers, 1 machinist, 1 carpenter	,, 4.00	=	16.00	
1 blacksmith	,, 4.00	=	4.00	
1 ,, helper	,, 2.00	=	2.00	
1 teamster and Co. team, and 1 assayer	,, 4.00	=	8.00	
1 on tailing pit, and 3 roustabouts ...	,, 2.50	=	10.00	
15 men				$59.00

SUPPLIES.

Wood, Cords.

Dryers and furnace, 12 cords	@ $4.50	= $54.00	
Boilers, 5 cords	,, 4.50	= 22.50	
Hauling same	,, 0.40	= 6.80	
17 cords			$83.30

Chemicals.

325 lb. sulphur	@ .01¼	= $4.06	
200 ,, hyposulphite	,, .06½	= 13.00	
175 ,, caustic soda	,, .05	= 8.75	
300 ,, blue stone	,, .07¼	= 21.75	
1000			$47.56

Salt.

5 tons	@ $26.00	= $130.00	
			$130.00
Incidentals, oils, assay materials, repairs			$12.00

GENERAL EXPENSE.

Office management, &c.	$27.00
Total	$457.86

EXPENSES PER TON.

Labour	= $1.61
Supplies	= 2.73
General expense	= .27
Total cost per ton	$4.61

In the estimates of expense given above, calcium hyposulphite is included. This reagent is used at first, simply to test the question of the solution of gold as compared with sodium hyposulphite. The latter, however, is eventually to be used. In the estimates for chlorurising and leaching, caustic soda is given and not soda ash. Both of these will be used in the future, but as the value of the lead carbonate extracted is equal to the amount of soda ash per ton, this has been omitted. When the lead is thrown down with soda ash less sodium sulphide is used, and, what is much more important, the sulphides are free from lead and can be made at once into silver bullion.

As compared with fusion or amalgamation processes this process offers all the advantages of a lixiviation plant, and in addition the possibility of making fine bullion free from lead, with a much higher extraction of the precious metals so far as the preliminary experiments, which have been made on a large

scale, can show. Its chief advantage lies not so much in the extraction of the precious metals from ores which are ordinarily leached, for although carefully conducted operations always give a higher extraction by the Russell process, the difference is but slight in some cases, as it does in the treatment of ores which would not ordinarily be leached, and which can be treated with only an ordinary roasting or perhaps with no roasting at all.

This process requires much less capital than Pan amalgamation, and the reagents used are far less expensive. It requires also less careful chlorurising roasting, the same time giving a much more complete extraction of the precious metal. With the limited experience so far, it appears that the value of the mercury lost in amalgamation very much exceeds the expense of waste of chemicals in the Russell process, which are not injurious to health, while mercury is, and allows at the same time the extraction of lead and copper as secondary products which are entirely lost by amalgamation.

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END OF VOL I.

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ERRATA.

Page 43. 17th line from the bottom—for "Crooks in 1858," read Parks about 1850.

Page 76. 3rd line from the top—for "$10," read about $6.50.

Page 117. Foot note. These works were transferred to Argo, near Denver, in 1878.

Page 516. 10th line from the top—for "30," read 20: for "40," read 30.

Foot note to the same page. In some of the Montana mills No. 15 screens were being used in amalgamation in August, 1887.